UNIVERSITY OF TORONTO
3 1761 00352948 4

Engineering Library
This volume is damaged or brittle
and CANNOT be repaired
Handle with EXTREME CARE

DEPARTMENT OF [illegible]
126 MES

Section - C3
No - 152

THE

LIXIVIATION OF SILVER-ORES

WITH

HYPOSULPHITE SOLUTIONS,

WITH

SPECIAL REFERENCE TO THE RUSSELL PROCESS.

BY

CARL A. STETEFELDT.

97413
6/8/09

SECOND EDITION.

Published, and sold in the United States and Mexico, by the Author.

For all other countries the sale is conducted by

Craz & Gerlach (Joh. Stettner), Freiberg, in Sachsen, Germany.

TN
770
S84

COPYRIGHT, 1895
BY CARL A. STETEFELDT.

THE TUTTLE MOREHOUSE & TAYLOR PRESS, NEW HAVEN, CONN.

TO HIS

TEACHER AND FRIEND,

GEHEIMER BERGRATH, PROFESSOR BRUNO KERL,

of Berlin,

THIS TREATISE IS DEDICATED

BY

HIS GRATEFUL PUPIL,

THE AUTHOR.

PREFACE.

In this second edition of the Lixiviation of Silver-Ores with Hyposulphite Solutions, some errors, which occurred in the first one, have been corrected, irrelevant matter has been left out, and a great deal of new matter has been added. How materially this new edition has been changed becomes evident from the fact that nearly one hundred pages of the text in the first edition have been cut out and replaced by new matter. I have not considered it necessary to reprint drawings of the details of a lixiviation-plant, such as ore-tanks, precipitating-tanks, etc., because all this apparatus is so simple that it can be described just as well.

A new plate, however, showing the arrangements of a modern lixiviation plant has been substituted for the old one in the first edition, and also a new drawing of a sulphide drier. I am under great obligations to Mr. Willard S. Morse, late Manager of the Aspen Mill, Colorado, for placing at my disposal numerous manuscripts recording investigations and experiments made by him and by his assistants, Stuart Croasdale and J. Dawson Hawkins. Such thorough and extensive work has not been done before or since in any metallurgical establishment in this country.

THE AUTHOR.

967 Alice Street,
Oakland, California,
May, 1895.

CONTENTS.

INTRODUCTION.

Page.
§ 1. History of the Lixiviation-Process 1
§ 2. Defects of the Old Lixiviation-Process 2
§ 3. The Russell Process 2
§ 4. Lixiviation Compared with Amalgamation 3
§ 5. History of the Russell Process 4
§ 6. Ores Suitable to Treatment by Lixiviation 6
§ 7. The Operations Preceding Lixiviation 7

PART I.

The Chemistry of the Lixiviation–Process.

CHAPTER I.

The Chemicals.

§ 101. Sodium Hyposulphite 9
§ 102. Calcium Hyposulphite 10
§ 103. Sodium Carbonate 10
§ 104. Caustic Soda 11
§ 105. Commercial Rating of Soda and Caustic Soda 11
§ 106. Sulphur 12
§ 107. Sulphuric Acid 12
§ 108. Crystals of Na_2S 12

The Chemical Reactions.

CHAPTER II.

THE REACTIONS OF SODIUM HYPOSULPHITE SOLUTIONS.

Page.
§ 201. Argentic Hyposulphite Double-Salts 14
§ 202. Double-Salts of Lead and Sodium Hyposulphite 14

SOLUBILITY OF METALS AND VARIOUS COMPOUNDS IN SODIUM HYPOSULPHITE SOLUTIONS.

§ 203. Solubility of Metallic Silver 15
§ 204. Solubility of Metallic Gold 15
§ 205. The Solubility of Silver and Gold in Sodium Hyposulphite Solutions Explained 16
§ 206. Solubility of Metallic Copper 16
§ 207. Solubility of Metallic Lead 17
§ 208. Solubility of Silver Oxide 17
§ 209. Solubility of Silver Sulphide 17
§ 210. Solubility of Gold Sulphide 17
§ 211. Solubility of Silver Arsenate 18
§ 212. Solubility of Silver Antimonate 18
§ 213. Practical importance of the Solubility of Silver Arsenate and Antimonate in Sodium Hyposulphite Solutions 18
§ 214. Solubility of Silver Chloride 19
§ 215. Influence of Lead Sulphate on the Solubility of Silver Chloride 20
§ 216. Influence of Calcium Sulphate on the Solubility of Silver Chloride 20
§ 217. Influence of Sodium Sulphate on the Solubility of Silver Chloride 21
§ 218. The Effect of Caustic Soda and Caustic Lime upon the Solubility of Silver-Compounds 21
§ 219. The Effect of a Caustic Hyposulphite-Solution Explained 22
§ 220. Solubility of the Carbonates of Silver, Copper, Lead, and other Metals 23
§ 221. Solubility of Lead Hydroxide and Cuprous Hydroxide 24
§ 222. Solubility of the Oxides and Hydroxides of Copper, Iron, Zinc, and Manganese 25
§ 223. Solubility of Cuprous Chloride 25
§ 224. Solubility of Lead Sulphate 25
§ 225. Solubility of Lead Antimonate 26
§ 226. Solubility of Calcium Sulphate 26
§ 227. Solubility of Caustic Lime 27
§ 228. Solubility of Sodium Antimonate 27

CHAPTER III.

THE REACTIONS OF EXTRA-SOLUTIONS.

Page.
§ 301. Cuprous Hyposulphite Double-Salts 28
§ 302. The Standard Extra-Solution 28

I. REACTIONS OF EXTRA-SOLUTIONS WITH CAUSTIC SODA AND SODIUM CARBONATE.

A. EXTRA-SOLUTION OF STANDARD COMPOSITION, CONTAINING FOR 1 PART $CuSO_4+5aq.$, 2¼ PARTS $Na_2S_2O_3+5aq.$

§ 303. Reactions with Caustic Soda 29
§ 304. Reactions with Sodium Carbonate 30

B. THE EXTRA-SOLUTION CONTAINS FOR 1 PART $CuSO_4+5aq.$, 4½ PARTS $Na_2S_2O_3+5aq.$

§ 305. Reactions with Caustic Soda 30
§ 306. Reactions with Sodium Carbonate 30

II. DECOMPOSITION OF EXTRA-SOLUTIONS BY HEATING.

§ 307. A. Extra-Solutions of Standard Composition 30
§ 308. B. Extra-Solutions containing for 1 part $CuSO_4+5aq.$, 4½ parts $Na_2S_2O_3+5aq.$ 31

III. DECOMPOSITION OF EXTRA SOLUTIONS AT ORDINARY TEMPERATURE.

A. EXTRA-SOLUTIONS OF STANDARD COMPOSITION.

§ 309. Solutions without free Acid 31
§ 310. Acid Solutions 32
§ 311. Solutions to which Sodium Carbonate has been added 32

B. EXTRA-SOLUTIONS CONTAINING FOR 1 PART $CuSO_4+5aq.$, 4½ PARTS $Na_2S_2O_3+5aq.$

§ 312. Solutions without free Acid 33
§ 313. Acid Solutions 33
§ 314. Solutions to which Sodium Carbonate has been added 33
§ 315. Solutions to which Caustic Soda has been added 34
§ 316. The Difference in Reactions Explained 34

Page.
§ 317. Conclusions of Practical Importance 34
§ 318. Investigation on the Formation of Sulphates in the Decomposition of Extra-Solutions 35
§ 319. Silver Tetrathionate 36
§ 320. The Effect of Extra-Solutions upon Silver-Compounds and Silver-Minerals, Compared with Amalgamation .. 36
§ 321. Extra-Solution with Cuprous Chloride 37

SOLUBILITIES OF SILVER, GOLD, LEAD, SILVER SULPHIDE, AND GOLD SULPHIDE IN EXTRA-SOLUTION.

§ 322. Solubility of Metallic Silver 37
§ 323. Solubility of Metallic Gold 38
§ 324. Solubility of Metallic Lead 38
§ 325. Solubility of Silver Sulphide 39
§ 326. Solubility of Gold Sulphide 40
§ 327. Solubility of Silver Arsenate and Antimonate 40

THE EFFECT OF EXTRA-SOLUTION ON SILVER-ORES.

§ 328. The Effect of Extra-Solution on Raw Ores 40
§ 329. The Effect of Extra-Solution on Ores that have been Oxidized by Roasting 44

THE EFFECT OF EXTRA-SOLUTION ON ORES THAT HAVE BEEN SUBJECTED TO A CHLORIDIZING-ROASTING.

§ 330. A. Roasted Ores free from Caustic Lime 48
§ 331. B. Roasted Ores containing Caustic Lime 50
§ 332. The Effect of Extra-Solution on Tailings 51

CHAPTER IV.

THE EXTRACTION OF GOLD 52

CHAPTER V.

CHEMISTRY OF THE WASH-WATER.

Page.
§ 501. The Composition of Roasted Ores 54
§ 502. Necessity of Leaching with Water 55

A. SALTS EASILY SOLUBLE IN WATER.

§ 503. Solubility of Sodium Sulphate 55
§ 504. Solubility of Sodium Chloride 55
§ 505. Solubilty of Calcium Chloride 55
§ 506. Solubility of the Sulphates and Chlorides of Manganese Zinc, Copper ($CuCl_2$), Iron, Aluminum, and Magnesium 55

B. SALTS NOT FREELY SOLUBLE IN WATER.

§ 507. Solubilty of Cuprous Chloride 56
§ 508. " " Lead Chloride 56
§ 509. " " Calcium Sulphate 56
§ 510. " " Caustic Lime 56
§ 511. " " Sodium Antimonate 57

C. SALTS NOT EASILY SOLUBLE IN WATER, BUT MORE SOLUBLE IN BRINE.

§ 512. Solubility of Cuprous Chloride 57
§ 513. " " Lead Chloride 57
§ 514. " " Calcium Sulphate 57

D. SALTS PRACTICALLY INSOLUBLE IN WATER, BUT SOLUBLE IN BRINE AND IN SOLUTIONS OF OTHER CHLORIDES.

§ 515. Solubility of Silver Chloride in Brine 58
§ 516. Solubility of Silver Chloride in Solutions of various Chlorides 59

R. SALTS INSOLUBLE IN WATER, BUT SOLUBLE IN WASH-WATER BRINE.

§ 517. Solubility of Silver Chloride 60
§ 518. " " Silver Arsenate and Antimonate 61
§ 519. " " Lead Sulphate 61
§ 520. " " Lead Antimonate 61
§ 521. Conclusions 62
§ 522. Effect of the First Wash-Water on the Chlorination of the Ore, or the so-called " going back " of the Chlorinations 63

CHAPTER VI.

THE PRECIPITATION OF METALS FROM HYPOSULPHITE SOLUTIONS.

Page.
§ 601. Preparation of Sodium and Calcium Sulphides in the Mill 71
§ 602. Formation of Hyposulphite Salts 72
§ 603. Economy of Producing Hyposulphites by Oxidation of Sulphide Solutions 75
§ 604. Precipitation of Calcium Carbonate by Sodium Sulphide Solutions 77
§ 605. Regeneration of Sodium Hyposulphite from Tetrathionate 79
§ 606. Precipitating-Coefficients 81
§ 607. Application of the Tables to Special Cases 88
§ 608. The Loss of Hyposulphite in Lixiviation 90

CHAPTER VII.

LABORATORY WORK.

§ 701. Determination of the Percentage of Salt in Crushed Ore 94
§ 702. Determination of Soluble Salts in Roasted Ore 94
§ 703. Lixiviation-Test with Ordinary Solution 95
§ 704. Influence of re-pulverizing Roasted Ore in an Iron Mortar 95
§ 705. Influence of Copper Vessel 96

LIXIVIATION-TESTS WITH EXTRA-SOLUTION.

§ 706. Lixiviation-Tests with Extra Solution for Raw Ores and Tailings 97
§ 707. Lixiviation-Tests with Extra Solution for Roasted Ores and Tailings 98
§ 708. Control of the Mill-Work by Laboratory-Tests essential in Lixiviation 98
§ 709. Volumetric Determination of Sodium Hyposulphite with Iodine 99
§ 710. Influence of Silver, Copper, and Lead on the Correctness of Results 101
§ 711. Assaying of Ores, Tailings, and Lixiviation-Products 102
§ 712. Testing Ores regarding their Fitness for Lixiviation 103

PART II.

THE LIXIVIATION-PROCESS IN ITS PRACTICAL EXECUTION.

CHAPTER XI.

THE LIXIVIATION-PLANT.

Page.
§ 1101. Wooden Tanks 105
§ 1102. Precipitating-Tanks for Wash-Water 110
§ 1103. Sodium Sulphide Tanks 110
§ 1104. Apparatus for Heating the Stock-Solution 110
§ 1105. Apparatus for Raising Stock-Solution to the Storage-Tanks 111
§ 1106. Apparatus for Increasing the Rate of Lixiviation 112
§ 1107. Apparatus for Circulating Extra-Solution 113
§ 1108. Filter-Press and Press-Tank 114
§ 1109. Drying-Chamber for Precipitates 114
§ 1110. Auxiliary Machinery 115
§ 1111. Pipe-Lines 115
§ 1112. Cocks 116
§ 1113. Protection of Iron by Asphaltic Covering, and other Notes on Construction 116
§ 1114. General Remarks about Construction 117
§ 1115. Cost of Modern Lixiviation-Plants 118

CHAPTER XII.

THE SOLUTIONS.

§ 1201. Definition of Terms 121
§ 1202. The Ordinary or Stock-Solution 121
§ 1203. The Extra-Solution 124
§ 1204. The Sodium Carbonate Solution 127
§ 1205. Caustic Lime 127
§ 1206. The Sodium and Calcium Sulphide Solutions 127

CHAPTER XIII.

THE CHARGING AND DISCHARGING OF THE ORE-TANKS.—GENERAL RULES FOR HANDLING THE LIXIVIATION-SOLUTIONS.

§ 1301. Charging the Ore-Tanks 128
§ 1302. General Rules for Handling the Lixiviation-Solutions 129
§ 1303. Discharging of the Tailings and their Sampling 133

CHAPTER XIV.

THE TREATMENT OF ROASTED AND RAW ORES.

Page.
§ 1401. The First Wash-Water 135

TREATMENT OF THE SILVER-BEARING WASH-WATER.

§ 1402. Extraction of Silver from Wash-Water by Precipitation with Sodium or Calcium Sulphide 142
§ 1403. Extraction of Silver from Wash-Water by Precipitation with Copper or Iron 142

THE LIXIVIATION.

§ 1404. Lixiviation of Raw Ores 143
§ 1405. Lixiviation of Acid Roasted Ores 144
§ 1406. Lixiviation of Arsenical Alkaline Roasted Ores 145
§ 1407. Lixiviation of Alkaline Roasted Ores 145
§ 1408. Distribution of Silver and Lead in the Solution during different Periods of Lixiviation 146
§ 1409. Method of Determining whether the Extraction of the Silver is completed 148

CHAPTER XV.

TROUGH LIXIVIATION 149

CHAPTER XVI.

THE PRECIPITATION OF THE METALS FROM A LIXIVIATION-SOLUTION IN THE MILL.

§ 1601. Constitution of a Lixiviation-Solution before Precipitation 153
§ 1602. The Precipitation of Lead [and Calcium] with Sodium Carbonate 153
§ 1603. The Precipitation of Silver, Gold, Copper [and Lead]. 154
§ 1604. Mechanical Handling of the Sulphides 155

CHAPTER XVII.

THE REFINING OF SULPHIDES.

§ 1701. The Old Practice 157

REFINING BY HUMID PROCESSES.

Page.
§ 1702. The Stetefeldt Process 158
§ 1703. The Sulphuric Acid Process 179

CHAPTER XVIII.

THE RUSSELL PROCESS COMPARED WITH ORDINARY LIXIVIATION, AND WITH AMALGAMATION.

A. THE RUSSELL PROCESS COMPARED WITH ORDINARY LIXIVIATION.

§ 1801. Results at the Cusihuiriachic Mill, Chihuahua, Mexico. 182
§ 1802. Results at the Sierra Grande Mill, New Mexico 182
§ 1803. Results at the Yedras Mill, Sinaloa, Mexico 183

B. THE RUSSELL PROCESS COMPARED WITH AMALGAMATION.

§ 1804. Results at the Sierra Grande, Cusihuiriachic, and Blue Bird Mills 187
§ 1805. Results at the Marsac and Ontario Mills, Park City, Utah, on Daly Ore, obtained in 1893–94 189

CHAPTER XIX.

LIXIVIATION AT THE MARSAC, ASPEN AND MOUNT CORY MILLS.

§ 1901. Lixiviation at the Marsac Mill, Park City, Utah 191
§ 1902. Lixiviation at the Aspen Mill, Colorado 195
§ 1903. Lixiviation at the Mount Cory Mill, Nevada 200

THE
LIXIVIATION OF SILVER-ORES
WITH
HYPOSULPHITE SOLUTIONS,
WITH SPECIAL REFERENCE TO THE
RUSSELL PROCESS.

INTRODUCTION.

The extraction of silver by the lixiviation-process from raw ores, and from ores after they have been subjected to a chloridizing-roasting, is based upon the fact that silver chloride, antimonate and arsenate, and, to a certain extent, even metallic silver, are soluble in solutions of sodium or calcium hyposulphite, and that silver sulphide is precipitated from such solutions by sodium or calcium sulphide. In case the ore contains lead, a portion of the latter is also dissolved, lead sulphate being soluble in hyposulphite solutions. If, at the same time, copper is present in roasted ore in the form of cuprous chloride, the sulphides precipitated from the lixiviation-solution contain silver, copper, and lead, besides small amounts of other metals. From gold-bearing silver-ores, gold is obtained together with the silver, but the percentage of its extraction varies, and depends upon many circumstances. Only ores of low value in gold, say below one ounce per ton, yield a high percentage of this metal.

§ 1. *History of the Lixiviation-Process.*

This process was first introduced by von Patera, an Austrian metallurgist, at Joachimsthal, in 1858. Von Patera used sodium hyposulphite, for which Kiss substituted the calcium-salt, in 1860, at Schmoellnitz. In the United States, the process was first employed by Kuestel, in 1874, at Melrose, California, and later, principally by O. Hofmann, it was introduced in several mills in California, Nevada, and Arizona, none of which are, however, in operation at present. The use of the process was revived by the construction of the Bertrand mill, in 1882, the Mount Cory mill, in 1883, and the Cortez mill, in 1885, all in Nevada.

§ 2. *Defects of the Old Lixiviation-Process.*

Although lixiviation is cheaper than amalgamation, both as to cost of plant and working-expense per ton of ore, there were many reasons why its adoption made slow progress. Leaving out the fact that there always exists a prejudice against any new method that requires more skill and chemical knowledge than those in vogue, the lixiviation-process, as executed by von Patera, Kiss, Kuestel, and Hofmann, had some defects which made it inapplicable to the treatment of certain ores.

A high chlorination of the silver cannot always be obtained by roasting, especially in case the ore contains calc-spar, which is converted, in part, into caustic lime. The caustic lime not only reduces silver chloride to metallic silver, but has also a deleterious effect on the solubility of most silver-compounds in hyposulphite solutions.

§ 3. *The Russell Process.*

These defects have been overcome by the Russell process, so named after its inventor, E. H. Russell.

Russell discovered that a solution of a double-salt of sodium and cuprous hyposulphite, formed by mixing sodium hyposulphite with copper sulphate, exerts a most energetic dissolving and decomposing action upon metallic silver, silver sulphide, silver-minerals belonging to the group of antimonial and arsenical sulphides, and other silver-combinations. Hence, if a charge of ore is first lixiviated with ordinary sodium hyposulphite solution, and, subsequently, with a solution containing a double-salt of sodium and cuprous hyposulphite,—this solvent is called the extra-solution—an additional amount of silver is extracted which would have been lost in the tailings by working according to the old method alone.

Russell also discovered that lead can be completely separated from a sodium hyposulphite solution, as lead carbonate, by sodium carbonate. After decanting the solution from the lead carbonate, silver and copper are precipitated from it in the usual way. This method of separating lead prohibits the use of calcium polysulphide as a precipitant for silver, because calcium entering the regenerated lixiviation-solution would also be precipitated as a carbonate with the lead by sodium carbonate. Hence, sodium sulphide must be employed. A full investigation has demonstrated that this is by no means detrimental. Sodium sulphide and hyposulphite are more advantageously used in the lixiviation-process than the corresponding calcium-salts.

§ 4. *Lixiviation compared with Amalgamation.*

In comparing lixiviation with amalgamation, the principal items in favor of lixiviation are:

1st. In amalgamating roasted ores, the coarseness of crushing is limited by the capacity of the settlers to work off coarse sands without loss of quicksilver. In lixiviation, pulverizing as coarse as possible is desirable. The limit of coarseness for roasting depends upon the character of the ore, and, principally, upon the manner in which the silver-bearing minerals are distributed in the gangue.

From this it follows that a given crushing-plant may have a much greater capacity for lixiviation than for amalgamation.

2d. The original cost of a lixiviation-plant is less than that of pans and settlers. A further saving is effected by a reduction in cost of the engine and boilers.

3d. In amalgamation the pans and settlers consume not less than 1⅓ horse-power per ton of ore. The power for pumping solutions, etc., in the lixiviation-process, is merely nominal.

4th. In large pan-mills the quantity of quicksilver in rotation represents a capital of from $20,000 to $25,000, while the stock of chemicals required for lixiviation costs about one-tenth of this amount.

5th. With Russell's improvements, the percentage of silver extracted by lixiviation is higher than by amalgamation.

6th. Lixiviation by Russell's process requires a less careful chloridizing-roasting, and, in consequence, a lower percentage of salt may be used in roasting.

7th. The value of the lost quicksilver, and cost in wear and tear of the pans and settlers, is generally greater than that of the chemicals consumed in the lixiviation-process.

8th. In case gold-bearing silver-ores have been roasted with salt, lixiviation extracts more gold than amalgamation.

The disadvantages of lixiviation, as compared with amalgamation, are:

1st. Lixiviation requires more chemical knowledge, and a more careful supervision of the operations.

2d. The handling of large volumes of solutions is a disadvantage common to all humid processes of this character.

3d. There is more danger of losing silver by careless manipulation, and by leakage of a badly constructed plant.

4th. In the lixiviation-process the precious metals are obtained in the form of sulphides. The conversion of the latter into bullion requires more skill, and is more expensive than the handling of amalgam.

§ 5. *History of the Russell Process.*

The superiority of the Russell process, as compared with ordinary lixiviation and with amalgamation, has given a new impetus to the introduction of lixiviation.

The Russell process was first successfully introduced, in September, 1884, at Silver Reef, Utah, for the raw lixiviation of tailings resulting from raw amalgamation.

In December, 1884, Russell attempted the raw lixiviation of tailings, at Bullionville, Nevada. These tailings—the result of raw amalgamation of oxidized ores from the mines of Pioche District, Nevada—had been re-worked by amalgamation, and also by concentration. Their contents in precious metals were from 8 to 12 ounces of silver, and about $2.50 of gold, per ton. A more unfavorable material, especially in a mechanical sense, could not have been selected, since repeated grinding in pans had reduced the ore almost completely to slimes. The result, both as to extraction of silver and manipulation, was a complete failure. After roasting the tailings in a Stetefeldt furnace, they lixiviated well, mechanically speaking; but here another obstacle was met. The composition of the tailings was such that, without the admixture of sulphuret-ores, high chlorinations of the silver could not be obtained; and the extra-solution did not correct this defect in roasting sufficiently. The failure of procuring sulphuret-ores at a reasonable cost; the expense of drying the tailings; the high price of fuel, salt, and labor at Bullionville, left no profit and caused the closing of the mill.

Somewhat better were the results in lixiviating tailings at the Flagler Reduction-Works, Silver City, New Mexico, in 1885. Here also tailings from raw amalgamation, mostly in the form of slimes, made filtering exceedingly slow and difficult. It should be noted that, in both cases, the Russell process was applied, as a last resort, to worthless material from which silver could not be extracted with profit by any other process.

In February, 1886, the Sierra Grande Co. put into operation a large lixiviation-plant, calculated to work 80 tons of ore per day. These works, situated at Lake Valley, New Mexico, were designed to treat ores after they had been subjected to a chloridizing-roasting. The short-sighted policy of the company caused the erection of a cheap and badly constructed plant, designed and built by inexperienced engineers. The mill, mechanically speaking, was a complete failure; the success of the Russell process, however, was fully established. The ore was of such a character that ordinary lixiviation extracted the silver very imperfectly. On account of the expense of running a badly constructed mill, the machinery of which was constantly breaking down, the low-grade ore extracted from the mines did not yield a profit, and operations were soon suspended, the company being without funds to reconstruct the mill.

At the Ontario mill, Park City, Utah, Russell experimented with a small plant, in 1883-84. W. A. Wilson commenced, with the same plant, a series of lixiviation-tests with Daly ore, in 1887, which led to the introduction of lixiviation, in place of amalgamation, at the Marsac mill, Park City, Utah, in 1888.

Ellsworth Daggett adopted the Russell process, in place of ordinary lixiviation and amalgamation, at the Cusihuiriachic mill, Chihuahua, Mexico, in 1886.

A brilliant success is recorded at the Yedras mill, Sinaloa, Mexico, where the Russell process was substituted for ordinary lixiviation, in November, 1887.

A lixiviation-plant, for the Russell process, was erected at the Blue Bird mill, Butte, Montana, in 1889. After the Blue Bird mine ceased to supply ore to the mill, tailings from amalgamation were treated with profit.

The Sombrerete mill, Mexico, was started in May, 1889, but closed again in the following year after treating about 5,000 tons of ore.

Another lixiviation-mill, especially designed for the Russell process, was built at Aspen, Colorado, in 1891.

The Bi-Metallic Mining Co., Phillipsburg, Montana, put in operation a Russell process plant, in 1894, for treating tailings from the amalgamation of roasted ores.

In spring of 1893, the construction of a large lixiviation-plant, especially designed for treating old tailings from amalgamation by the Russell process, was commenced at the Ontario mill, Park City, Utah. When the sudden decline in the price of silver took place, in June, 1893, construction was suspended after finishing the building, although most of the materials and machinery were on hand at the mill. Detailed estimates of cost on this plant are given in § 1,115.

§ 6. *Ores Suitable to Treatment by Lixiviation.*

It is safe to say that all dry silver-ores fit for amalgamation, after roasting, can also be treated by lixiviation, and that the latter process, with Russell's improvements, may succeed where ordinary lixiviation and amalgamation were failures.

Oxidized ores, containing silver chloride and lead-minerals, may be lixiviated after crushing, and the tailings concentrated for lead. This has been done at the Old Telegraph mine, Utah.

The Russell process is also well adapted to the treatment of tailings resulting from ores which have been worked either by the old lixiviation-process or by amalgamation.

In the lixiviation of raw ores, generally speaking, too little practical experience is extant to say whether the Russell process can compete, successfully, with raw amalgamation for the extraction of silver. Some laboratory-tests with raw ores are recorded in § 328.

§ 7. *The Operations Preceding Lixiviation.*

A detailed description of the operations preceding the lixiviation-process, namely: crushing, or crushing and roasting of the ore, is not necessary here, since these operations are identically the same as those preceding the well-known treatment by amalgamation.

As to crushing-machinery, I would give preference to well constructed rolls, like those furnished by S. R. Krom, in place of stamps. Pulverizing by rolls produces much less fine dust, and a minimum quantity of the latter facilitates the filtering of the solutions through the change in the ore-tanks, especially if the ore is treated raw. The size of the screen through which the crushed ore should be made to pass, in order to roast and lixiviate with most economical results, can only be determined by a practical test in each individual case, since it depends upon the character of the ore.

In dry-crushing pan-mills, where the ore is subjected to a choridizing-roasting, only stamps have been used so far. In such mills screens not coarser than No. 26 are generally used on the battery, the settlers being unable to do good work with coarser material. On the contrary, experiments are on record in which ore crushed through a No. 6 wire-screen was roasted and lixiviated successfully.

In wetting down roasted ore on the cooling-floor, a very injurious effect may be produced if this operation is performed while the ore is red-hot. The effect is most pronounced in case ordinary lixiviation is used, but the extraction is also lessened in treating the ore with extra-solution. Evidently, the steam, coming in contact with the ore at a high temperature, causes a decomposition of certain silver-compounds, otherwise soluble in hyposulphite solutions. In some cases it is best to leave the ore on the cooling-floor until it is sufficiently cold for charging the ore-tanks.

Tables Nos. 9., a. and b., illustrate what has been said above.

At the Aspen mill, Colorado, ore taken directly from the shaft of the Stetefeldt furnace was allowed to cool dry, and then thrown into a barrel containing sufficient water to saturate the ore.

Another portion was thrown red-hot into a barrel as above. Chlorination-tests were then made with ordinary and extra-solution.

TABLE No. 9. a.

Showing the Effect of Wetting Down Ore on the Cooling Floor while Red-Hot.

Description of Ore.	Value in Silver per Ton. Ounces.	Percent. Extraction in Assay-Office.			
		Ordin. Sol. Wetting down.		Extra-Sol. Wetting down.	
		Cold.	Hot.	Cold.	Hot.
San Bartolo, Cusi, Mexico. Acid......	49.1	69.1	20.3	86.3	80.3
Sierra Grande, Lake Valley, New Mexico. Alkaline............	16.0	81.5	62.5	87.5	71.9

The roasting of the ore was done in Howell and White furnaces.

TABLE No. 9. b.

Showing the Effect of Wetting Down Aspen Ore Cold and Red-Hot.

Ore Wetted Down:	Silver-Value, per Ton. Ounces.	Percent. of Silver extracted in Assay-Office.	
		Ord. Sol.	Extra-Sol.
Cold..............................	23.4	38.0	90.6
Red-Hot.........................	21.1	28.5	85.3

PART I.

THE CHEMISTRY
OF THE
LIXIVIATION-PROCESS.

CHAPTER I.

THE CHEMICALS.

The chemicals used in the lixiviation-process are: Sodium hyposulphite, for making the stock-solution; copper sulphate, for preparing Russell's extra-solution; sodium carbonate and caustic-lime, for precipitating lead; caustic soda, caustic lime, and sulphur, for preparing sodium or calcium sulphide; sulphuric acid, for neutralizing caustic lixiviation-solutions, for treatment of the wash-water, and for refining sulphides. Crystals of Na_2S are also a commercial article, and may be used for precipitating silver sulphide.

§ 101. *Sodium Hyposulphite.*

Sodium hyposulphite is the only hyposulphite salt that is manufactured on a large scale. It is principally imported from England and Germany. It has the formula $Na_2S_2O_3+5$ aq., and does not deteriorate in contact with the atmosphere. Its solutions are also quite permanent, and decompose slowly either at ordinary temperature or at boiling-point.

The solubility of sodium hyposulphite is as follows. 1000 c. c. water dissolve:

At 0° C.,	497.5 gm.	$Na_2S_2O_3+5$ aq.
" 20° C.,	694.4	" " "
" 40° C.,	1041.6	" " "
" 60° C.,	1923.0	" " "

Acids added to the solution precipitate sulphur, and evolve sulphurous acid. The crystals of $Na_2S_2O_3+5$ aq. melt at 56° C.; at 100° C., the water is completely driven out. Upon exposing the dry salt to a temperature approaching red-heat, it emits sulphur, and a yellow mass remains, consisting of sodium sulphide and sulphate.

§ 102. *Calcium Hyposulphite.*

Calcium hyposulphite, CaS_2O_3+6 aq., is not an article of commerce. It can only be made with difficulty in the laboratory, since its solution has to be evaporated at a temperature below 60° C. At a higher temperature decomposition takes place, sulphur and calcium sulphite, $CaSO_3$, being formed. Calcium hyposulphite solutions also deteriorate much quicker than those of the sodium-salt at ordinary temperature. Lixiviation on a large scale is now always commenced with a sodium hyposulphite solution. If then calcium sulphide is used as a precipitant for silver, the sodium hyposulphite is gradually replaced by the calcium-salt. A solution of calcium hyposulphite is best obtained by conducting sulphurous acid through a solution of calcium sulphide until the latter becomes colorless and neutral.

§ 103. *Sodium Carbonate.*

It is most economical to buy the soda made by the Solvay or ammonia process, and not the impure, common soda-ash. Solvay soda is manufactured in this country, at Syracuse, N. Y.

This alkali, in the form of a white powder, is so pure that it contains 98.7 per cent. of sodium carbonate, the remainder being, principally, sodium chloride and sulphate, and 0.25 per cent. moisture. On account of the entire absence of sodium sulphide, which is always found in the ordinary soda-ash, the solution for the precipitation of lead does not require purification.

The solubility of sodium carbonate is as follows. 1000 c. c. water dissolve:

At 0° C., 69.7 gm. Na_2CO_3.
“ 10° C., 120.9 “ “
“ 20° C., 217.1 “ “

§ 104. *Caustic Soda.*

Only a high-grade article should be ordered, of 70 to 76 per cent. It is put up in sheet-iron drums, holding 630 to 700 pounds. It should not be left exposed to the air, after a drum has been opened, as it absorbs carbonic acid.

The solubility of caustic soda is as follows. 1000 c. c. water dissolve:

At 15° C., 600 gm. NaHO.
" 100° C., 2100 gm. NaHO.

A saturated solution of caustic soda boils at 215.5° C. Caustic soda of high grade is manufactured by the Solvay Process Co., Syracuse, N. Y.

§ 105. *Commercial Rating of Soda and Caustic Soda.*

The manufacturers of soda-ash and soda rate their articles in a different way in different countries. In Germany, the percentage in Na_2CO_3 is given; in France, the percentage in Na_2O, and England follows the French custom, but substitutes 24 for the equivalent of Na, which is 23. This is one of the numerous mysteries of English commerce. Hence, the equivalent of Na_2O is taken at 64, and not at its true value of 62. The equivalent of Na_2CO_3 is taken at 108, in place of 106. Chemically pure soda, free from water, would be rated in Germany, at 100; in France, at 58.5; and in England, at 59.27 per cent. One French Gay-Lussac per cent. =1.013 English per cent.

The Solvay soda is generally rated at 58 per cent., and is consequently almost pure Na_2CO_3. The quotations of prices here, and in England, are based upon an article of 48 per cent., and purer qualities cost more in proportion to their percentage of Na_2O above 48 per cent. In regard to the valuation of soda-ash, it is necessary to mention that its valuation also includes the free NaHO, generally contained in it.

As to caustic soda, not its percentage in NaHO, but in Na_2O is given, as is the case with soda in France and England, and what has been stated about the change of the equivalent of Na in England, applies here also. Pure NaHO contains 77.5 per cent. Na_2O, but, according to false English conceptions, 78.04 per cent.

Quotations of caustic soda are based upon an article of 60 per cent. I give below analyses of some samples of English caustic soda.

Analyses of Caustic Soda.

NaHO	87.73 per cent.	87.50 per cent.	85.71 per cent.
Na_2CO_3	3.45 "	3.68 "	5.48 "
Na_2SO_4	1.00 "	1.32 "	2.10 "
NaCl	7.23 "	6.89 "	5.60 "
H_2O	0.59 "	0.68 "	1.11 "

These samples would be rated at 68, 67.8, and 66.4 per cent., according to French and German standards; and at 68.9, 68.7, and 67.3 per cent., respectively, according to English custom. The best qualities of caustic soda in the market generally contain from 70 to 74 per cent. Na_2O, and none is manufactured above 76 per cent.

§ 106. *Sulphur.*

It is immaterial whether the sulphur is crude or refined, in lumps or flowers of sulphur. In the preparation of sodium sulphide, fine pulverizing of the sulphur is not necessary.

§ 107. *Sulphuric Acid.*

Sulphuric acid of 66° B. is best transported in sheet-iron tanks which hold about 1700 pounds. If shipped in this way, the freight is much less than in glass carboys. The cost of a sheet-iron tank is $6.50.

If large quantities of sulphuric acid are needed for refining sulphides, it is shipped in large, iron tank-cars holding from 10 to 20 tons.

§ 108. *Crystals of Na_2S.*

Crystals of sodium mono-sulphide, Na_2S, are a commercial article. Whether it is more economical to buy Na_2S for precipitating silver from hyposulphite solutions, or to make sodium polysulphide in the mill from caustic soda and sulphur, depends entirely on local conditions. The advantages of using Na_2S are: (1) The manufacture of sodium sulphide in the mill is avoided.

(2) The sulphides are of higher grade in silver, no free sulphur being precipitated by using Na_2S.

The disadvantages of using Na_2S are: (1) The commercial sodium sulphide does not contain any hyposulphite, and the consumption of the latter salt in the mill is thereby increased. (2) If a solution of Na_2S is oxidized by the atmosphere, caustic soda is formed besides hyposulphite:

$$2Na_2S+H_2O+4O=Na_2S_2O_3+2NaHO.$$

This subject will be more fully discussed in Chapter VI.

The Chemical Reactions.

In the following pages I have endeavored to give a concise statement of the reactions that are of practical value. Those who wish to put lixiviation-works into successful operation should make themselves thoroughly familiar with the facts recorded below. We have to deal here, principally, with questions of solubility.

The solubility of a substance in a liquid may be considered as a function of temperature, concentration of the liquid, and time of reaction, other conditions being equal. In regard to the solubility of certain combinations of silver, and other metals, in solutions of sodium hyposulphite, we may state here at once that temperature enters in so far as an important factor, as moderately warm solutions, say from 25° to 55° C., act, in most cases, more energetically, and dissolve in less time, than colder ones. A much higher temperature of the solution, however, should be avoided.

CHAPTER II.

The Reactions of Sodium Hyposulphite Solutions.

§ 201. *Argentic Hyphosulphite Double-Salts.*

Silver forms with the hyposulphites of sodium, potassium, and calcium two series of double-salts. The one has the formula $2RS_2O_3+Ag_2S_2O_3+3aq$, and the other, $RS_2O_3+Ag_2S_2O_3+2aq$.

Of these double-salts the former are easily soluble in water, and the latter much less so.

Gold forms soluble double-salts of the formula: $3RS_2O_3+Au_2S_2O_3+4aq$.

The solutions of the argentic double-salts are quite permanent, and if neutral or alkaline, can be heated to boiling-point without injury. The addition of a very slight quantity of acid to a boiling solution, however, starts decomposition, silver being precipitated as sulphide. A slightly acid solution keeps at ordinary temperature; it commences to decompose if heated above 50° C.

The argentic hyposulphites differ materially in their reactions from other silver-salts. The most important differences are as follows, viz:

Chlorides, or hydrochloric acid, do not precipitate silver chloride.

Caustic soda does not precipitate silver oxide.

Sodium carbonate does not precipitate silver carbonate.

Solutions of potassium arsenate and antimonate do not precipitate silver arsenate and antimonate. The granular, white precipitate, which appears if potassium antimonate is added to the argentic sodium-salt, is sodium antimonate.

Ferrous sulphate does not precipitate metallic silver, either at ordinary temperature or upon heating.

§ 202. *Double-Salts of Lead and Sodium Hyposulphite.*

Lead forms a sodium double-salt of the formula: $2Na_2S_2O_3$, PbS_2O_3, which is soluble in an excess of sodium hyposulphite. If this solution is heated to about 70° C., it turns slightly dark. Upon boiling for some time, further decomposition does not seem to take place. A solution to which sulphuric acid has been added, becomes milky from separated sulphur, and turns dark at about

50° C. Upon boiling, decomposition takes place, lead sulphide being precipitated. Complete decomposition, however, is slow, lead being still found in solution after continued boiling.

Sulphuric acid and sulphates do not precipitate lead sulphate from solutions of lead hyposulphite.

In regard to caustics and carbonates of the alkalies, the reactions are the same as those of other lead-salts. The precipitation with sodium carbonate is very complete. Caustic soda precipitates a basic hydroxide of lead. The latter is soluble in caustic soda; hence, if an excess of the reagent is added, lead remains in solution. From the filtrate, sodium carbonate does not precipitate lead carbonate, the latter being also soluble in caustic soda.

The characteristics of the cuprous hyposulphites will be discussed in the next chapter.

SOLUBILITY OF METALS AND VARIOUS COMPOUNDS IN SODIUM HYPOSULPHITE SOLUTIONS.

§ 203. *Solubility of Metallic Silver.*

[In all the following experiments the metric system is used.]

0.500 gm. cement-silver were treated cold (at about 15° C.) for forty-eight hours with 1000 c. c. of solutions, containing respectively 10, 25, 50, 100, 150, 200, and 250 gm. $Na_2S_2O_3+5aq$. The result was that the 1 per cent. solution dissolved as much silver as the 25 per cent. solution, namely, 0.030 gm. Solutions of intermediate strength gave figures somewhat higher or lower. This shows that the dissolving energy of the solution is not increased by concentration. In repeating this experiment at a temperature of 50° C., the solubility of the silver was materially increased. The 20 per cent., 15 per cent., and 5 per cent. solutions dissolved 0.100 gm. silver each, and the others somewhat less. (Russell.)

§ 204. *Solubility of Metallic Gold.*

0.050 gm. gold-leaf were treated for forty-eight hours with 1000 c. c. of solutions of different concentration, as in § 203. In every instance about 0.002 gm. gold were dissolved, the deviations from this quantity being exceedingly slight. The same results were obtained with cement-gold. (Russell.)

§ 205. *The Solubility of Silver and Gold in Sodium Hyposulphite Solutions Explained.*

We have to seek the reason why concentration of the solutions does not increase the solubility of the precious metals. In order to form hyposulphite double-salts, the precious metals have to become oxidized, the oxygen being derived from the air absorbed by the solution. Hence, it is the quantity of free oxygen in the solution by which the result is, to a great extent, determined. Gold, being more difficult to oxidize than silver, is affected in a less degree. That elevation of temperature should materially favor the oxidation of the silver is not surprising. For one equivalent silver dissolved, one equivalent caustic soda is set free. This is demonstrated by the fact that a neutral solution becomes caustic as soon as silver has been taken up. The quantity of silver dissolved depends upon many circumstances, but is, principally, a function of time. Leaving 200 c. c. of a 2 per cent. sodium hyposulphite solution in contact with 4 gm. cement-silver for 3 days, at a temperature of about 20° C., I found 0.023 gm. silver dissolved. The same experiment carried out in a stoppered flask, excluding access of air, and boiling the solution before adding the silver, gave only a trace of silver in solution.

§ 206. *Solubility of Metallic Copper.*

The solution of copper is effected in the same way as that of silver, both cuprous hydroxide and cuprous carbonate being soluble in sodium hyposulphite. The quantity of copper dissolved is, principally, a function of time and temperature.

200 c. c. of a solution containing 2 per cent. $Na_2S_2O_3+5aq.$, left in contact with cement-copper, dissolved:

At about	15° C., in 3 days,	0.079 gm. Cu.
"	20° C., in 24 hours,	0.030 "
"	50° C., in 2 hours,	0.011 "

If a sodium hyposulphite solution is heated to boiling in contact with copper, it dissolves much less of that metal than at a moderately warm temperature, and a precipitate of Cu_2S appears after some time. The reaction will be explained in § 308.

§ 207. *Solubility of Metallic Lead.*

A sodium hyposulphite solution, exposed to contact with the atmosphere, does not dissolve metallic lead. This is based upon the fact that both lead hydroxide and carbonate are insoluble in hyposulphite salts.

§ 208. *Solubility of Silver Oxide.*

Ag_2O dissolves in a sodium hyposulphite solution, setting NaHO free. Cold solutions act with less energy than warm ones. Silver is dissolved in proportion to the quantity of sodium hyposulphite present. The solubility-coefficient, under most favorable circumstances, is about 0.200 Ag per unit of $Na_2S_2O_3+5aq$. (Russell.) According to the formula:

$$3Na_2S_2O_3+Ag_2O+H_2O=2Na_2S_2O_3,\ Ag_2S_2O_3+2NaHO,$$

at least a solubility-coefficient of 0.289 should be expected. Evidently, the presence of free NaHO either prevents a complete reaction, according to the above formula, or a caustic solution is a poorer solvent for the argentic double-salts than water. The deleterious influence of caustics on the solubility of silver in hyposulphite solutions is observed throughout, with the exception of a few cases only, as will be demonstrated later.

§ 209. *Solubility of Silver Sulphide.*

Silver was not dissolved after exposing 2 gm. Ag_2S to sodium hyposulphite solutions of different concentration for twenty-four hours.

§ 210. *Solubility of Gold Sulphide.*

Gold sulphide is decomposed by a solution of sodium hyposulphite. For energetic action, heating to 50° or 60° C. is necessary. In treating 2 gm. Au_2S_3 for twenty-four hours cold, 0.066 gm. gold were dissolved; while at a temperature of 65° C., 0.117 gm. gold went into solution in two hours. An alkaline sulphide again precipitates Au_2S_3 from this solution. (Russell.)

In explanation of these results we have to consider that gold sulphide is easily decomposed to metallic gold and sulphur.

According to Levol, sulphuretted hydrogen precipitates from a boiling solution of gold chloride metallic gold, not Au_2S_3. The gold sulphide treated in the first experiment, most likely, contained metallic gold; and in the second experiment more metallic gold was formed. This, being in a state of exceedingly fine division, was much easier oxidized and dissolved than gold-leaf or cement-gold, used in the experiment of § 204. That Au_2S_3, as such, is not any more affected by a sodium hyposulphite solution than Ag_2S, follows from the fact that gold is again completely precipitated by alkaline sulphides.

§ 211. *Solubility of Silver Arsenate.*

Russell's results on the solubility of Ag_3AsO_4 are rather irregular, but sufficiently clear to base important conclusions upon them for practical purposes. Moderately warm solutions are much better solvents than cold ones. The solubility-coefficient found for Ag_3AsO_4 was, under most favorable circumstances, about 0.200 Ag per unit of $Na_2S_2O_3 + 5$ aq. The presence of caustic soda in a sodium hyposulphite solution materially increases the solubility-coefficient.

§ 212. *Solubility of Silver Antimonate.*

What has been said above, about irregularity of results, holds good in the case of $AgSbO_3$. Moderately warm solutions are decidedly better solvents than cold ones. The solubility-coefficient for warm solutions is about 0.050 Ag. per unit of $Na_2S_2O_3 + 5$aq. The presence of caustic soda increases the solubility of $AgSbO_3$. The effect, however, is by no means so marked as in the case of silver arsenate. The much inferior solubility of silver antimonate, as compared with the arsenate, is explained by the fact that sodium arsenate is much more soluble than the antimonate, these salts being formed in the reactions.

§ 213. *Practical Importance of the Solubility of Silver Arsenate and Antimonate in Sodium Hyposulphite Solutions.*

I pause to point out the practical importance of the facts recorded in the last two paragraphs. The majority of silver-ores, that are subjected to a chloridizing-roasting, contain antimony, or arsenic, or both. Although in chloridizing-roasting antimony and arsenic are mostly changed to chlorides, and pass off with the roasting-gases, the formation of silver antimonate and arsenate cannot be entirely prevented, these salts not being completely decomposed in contact with sodium chloride in the roasted ore. In treating the ore by lixiviation, the presence of these silver-salts is not detrimental, since they are amenable to extraction by hyposulphite solutions. In amalgamation, on the contrary, they remain mostly undecomposed, and this is the principal reason why hyposulphite solutions extract silver from tailings resulting from amalgamation of roasted ores. I have observed that the percentage of silver extracted by lixiviation from such tailings is greatly lessened if the roasting is carried on at a very high temperature. This agrees with Plattner's observation that in fusing silver arsenate or antimonate with sodium chloride, silver chloride and the respective sodium-salts are formed.

§ 214. *Solubility of Silver Chloride.*

In dissolving AgCl by sodium hyposulphite, both double-salts mentioned at the beginning of this chapter are formed.

Two methods can be used in determining the solvent energy of sodium hyposulphite for silver chloride.

1st. By exposing freshly precipitated AgCl to the solution.

2d. By adding $AgNO_3$ to a hyposulphite solution which contains sodium chloride. As soon as saturation has taken place the solution become turbid.

The first method gives somewhat varying results if a large surplus of AgCl is used, and left exposed to the solution for a long time. Then, the less soluble double-salt, $Ag_2S_2O_3$, $Na_2S_2O_3 + 2aq.$, is formed. By using solutions of more than 10 per cent. concentration, the AgCl hardens into lumps, because in such cases the quantity of water present is not sufficient to dissolve the argentic double-salts. For solutions of lower concentration, the solvent energy is almost entirely a function of concentration, and increases in proportion to the quantity of sodium hyposulphite present. Temperature does not influence results materially.

Russell's determinations, by the first method, give an average coefficient of 0.301 Ag, or 0.400 AgCl, for one part of $Na_2S_2O_3 + 5aq.$

By the second method, the coefficient is 0.365 Ag, or 0.485 AgCl, for one part of $Na_2S_2O_3+5aq$.

It is of considerable interest to know to what degree the solubility of AgCl is influenced if other soluble salts are mixed with the solution, or if the latter has been saturated with salts not very soluble in water, but soluble in sodium hyposulphite. The principal combinations which in this respect play an important part in the lixiviation-process are the sulphates of lead, calcium and sodium, caustic soda, and caustic lime; and these alone we shall consider.

§ 215. *Influence of Lead Sulphate on the Solubility of Silver Chloride.*

If lead sulphate has been dissolved in a sodium hyposulphite solution, and the latter is tested for its solvent energy for silver chloride, by the first method, the results show a marked decrease. It is not astonishing that this should be so, because a part of the sodium hyposulphite is consumed in the formation of the lead-salt. Assuming that this has the formula $2Na_2S_2O_3, PbS_2O_3$, and calculating the amount of sodium hyposulphite necessary in its formation, we find that the dissolving energy of the solution for silver chloride has been less reduced than should be expected, taking Russell's value of 0.400 for the solubility of silver chloride as a basis for calculation. From Russell's experiments I have deduced the following table:

Solution contains for one part $Na_2S_2O_3+5aq$. dissolved parts of $PbSO_4$.	Decrease of dissolving energy for AgCl.	Decrease expected by the formation of $2Na_2S_2O_3, PbS_2O_3$.
0.045 parts.	8.6 per cent.	11.0 per cent.
0.054 "	11.2 "	13.2 "
0.059 "	12.6 "	14.4 "
0.086 "	15.4 "	21.0 "
0.107 "	23.8 "	26.2 "

§ 216. *Influence of Calcium Sulphate on the Solubility of Silver Chloride.*

In testing the solubility of silver chloride in a sodium hyposulphite solution saturated with gypsum, the decrease of solvent energy was very slight. The reason is obvious. A double-salt of sodium and calcium hyposulphite is formed, and sodium sulphate. Decrease in solvent energy would only be produced by the last-named salt, the formation of which is very limited in this case. (See § 226.)

§ 217. *Influence of Sodium Sulphate on the Solubility of Silver Chloride.*

The presence of this salt depresses the solvent energy of sodium hyposulphite for silver chloride. In experimenting with solutions containing 1½ per cent. of $Na_2S_2O_3+5aq.$, their solvent energy was diminished 6 per cent. by addition of 2 per cent. of sodium sulphate, and 8 per cent. by addition of 5 per cent. of this salt.

§ 218. *The Effect of Caustic Soda and Caustic Lime upon the Solubility of Silver-Compounds.*

Russell found that the presence of a very slight quantity of caustic soda or lime in a hyposulphite solution has a very deleterious effect on the extraction of silver chloride and other silver-combinations from ores. In adding one-half of one per cent. of caustic lime to a sodium hyposulphite solution, the percentage of silver extracted from Ontario ore was depressed from 11 to 24 per cent.

A sample of roasted Ontario ore, which yielded by lixiviation-tests with ordinary solution 86.5 per cent. of its silver, was first leached with water, and then treated for thirty-six hours with sodium hyposulphite solutions of 2½ per cent. concentration to which various amounts of caustic soda had been added.

Per cent. of caustic soda added.	Per cent. of the silver extracted.
0.2	6.5
0.5	3.7
1.0	4.8
5.0	5.9

We note here that, after the maximum effect has been produced, an increase of caustic soda extracts more silver. This result is due to the increased solubility of silver arsenate and antimonate in a caustic solution. A sodium hyposulphite solution of 2½ per cent. concentration, saturated with caustic lime, extracted from the same ore 60 per cent. of its silver. In case lixiviation-tests were made without first leaching with water, the effect of caustics was materially lessened by their action upon the sulphates and chlorides of copper, zinc, and manganese.

From numerous experiments it appears that the effect of caustic soda is not so pronounced with all ores as in the example given above.

A lixiviation-solution may become caustic:

1st. If roasted ores containing caustic lime are treated.

2d. In consequence of the presence of free caustic in sodium sulphide used for precipitation of the silver.

3d. If lead is precipitated as basic hydroxide by caustic lime, and a surplus of the reagent has been added.

The solvent energy of a caustic lixiviation-solution is restored by neutralizing it with sulphuric acid. The same effect is produced by the addition of copper sulphate.

§ 219. *The Effect of a Caustic Hyposulphite Solution Explained.*

The deleterious effect of a caustic solution in the amalgamation of roasted ores is well known, and forms one of several parallels we meet in the processes of amalgamation and lixiviation.

It seems, in the first place, that caustics weaken the affinity of silver to form hyposulphite double-salts. Then, the silver choride in roasted ores containing lead is evidently more or less fused together with lead sulphate and chloride. A caustic hyposulphite solution converts the latter compounds into insoluble lead hydroxide and thus, in a purely mechanical way, may prevent complete solution of the silver chloride. If this theory is correct, the deleterious effect of caustics ought to increase the more lead the ore contains, and become very slight with ores free from lead. This has not been determined experimentally; I can only say that the sample of Ontario ore, mentioned in § 218, contained considerable lead.

The same question will be considered from a different standpoint in § 522.

§ 220. *Solubility of the Carbonates of Silver, Copper, Lead, and other Metals.*

Lead carbonate is insoluble in a sodium hyposulphite solution, while silver carbonate and cuprous carbonate are soluble. Hence, if sodium carbonate is added to a hyposulphite solution containing lead, silver, and copper, $PbCO_3$ alone is precipitated.* If a solution of $PbSO_4$ in sodium hyposulphite is so treated, the precipitation of the lead is so complete that H_2S gives no reaction in the filtrate. Cupric carbonate is also insoluble, but since cupric hyposulphites do not exist, a precipitation of the cupric salt cannot take place. Upon these reactions is based the separation of lead from silver and copper in Russell's lixiviation-process. In effecting this on a large scale, it is most economical to use the pure soda manufactured by the Solvay or ammonia process. If ordinary soda-ash is used, which contains more or less Na_2S, the solution must be purified, otherwise Ag_2S would be precipitated with $PbCO_3$. This is effected by adding to the soda-ash solution the so-called extra-solution (see next chapter), until a precipitate of copper sulphide ceases to appear. An excess of the reagent is not injurious.

The carbonates of iron, manganese, zinc, and calcium are also insoluble in a sodium hyposulphite solution. In well roasted ores only traces of iron-salts exist, and these, as well as the chlorides and sulphates of zinc and manganese, are removed by the first wash-water. Hence, none of these carbonates are precipitated with the lead carbonate, at least not in a perceptible degree.

If lead is to be precipitated by this process, the use of calcium sulphide is not admissible for precipitation of the silver.

Should roasted ore contain calcium sulphate and caustic lime in such quantities that their complete removal by the first wash-water is impracticable, then the lixiviation-solution will hold calcium-salts, and calcium carbonate will be precipitated together with the lead in attempting to separate this metal.

In this case the precipitate is always rich in silver.

If a sodium hyposulphite solution is impregnated with a sufficient quantity of sodium carbonate, prior to its application to lead-bearing ores, lead does not go into solution, but remains as a carbonate in the tailings.

---

* This, as will be seen later, is not absolutely true in practice. The lead-precipitate always contains some silver and may contain copper.

In experimenting with this process, Russell found that if sodium carbonate is added at once to the lixiviation-solution to prevent the extraction of lead, a much larger quantity of the reagent is consumed than in precipitating the lead after lixiviation is completed. This is explained by the fact that a dilute sodium hyposulphite solution will dissolve only a limited amount of lead sulphate. If, at the same time, sodium carbonate is present, the dissolving and precipitating actions of sodium hyposulphite and sodium carbonate go on indefinitely until the larger portion, or all of the lead sulphate in the ore, has been converted into carbonate. The same statement holds good in regard to calcium.

If to a neutral sodium hyposulphite solution, containing calcium-salts, a concentrated solution of sodium bi-carbonate is added, a precipitate of calcium bi-carbonate, $CaH_2(CO_3)_2$, is formed, which readily dissolves again upon stirring. A slight turbidity, however, remains. This is due to the presence of one or two per cent. of Na_2CO_3 in the commercial sodium bi-carbonate, whereby $CaCO_3$ is precipitated. From the filtered clear solution caustic soda throws down a copious precipitate of $CaCO_3$.

The reaction appears to be so delicate that it may be utilized in determining the percentage of Na_2CO_3 in $NaHCO_3$.

If a hyposulphite solution contains both lead and calcium, it becomes possible to separate lead carbonate by sodium bi-carbonate, retaining calcium in solution. Calcium carbonate would only be precipitated in case the solution was caustic.

The high price of sodium bi-carbonate is, however, against the application of this method in practice.

§ 221. *Solubility of Lead Hydroxide and Cuprous Hydroxide.*

Hydroxide of lead is insoluble, while cuprous hydroxide and silver oxide are soluble in hyposulphite solutions. Upon this is based the separation of lead from copper and silver by caustic lime. The separation of lead from silver by caustic lime is incomplete. According to C. Rueger, the lead-precipitate resulting at the Mount Cory mill, Nevada, contained 60 per cent. lead and 420 ounces, and more, silver per ton, besides a considerable amount of gypsum, and impurities of the lime.

This method should be used where a hyposulphite solution contains a certain amount of calcium, making precipitation of lead by sodium carbonate unprofitable. It is, of course, the only method for separate precipitation of lead where silver is precipitated by calcium sulphide.

§ 222. *Solubility of the Oxides and Hydroxides of Copper, Iron, Zinc, and Manganese.*

Cupric oxide and hydroxide, the oxides and hydroxides of iron, manganese, and zinc, are all insoluble in hyposulphite solutions.

§ 223. *Solubility of Cuprous Chloride.*

Cuprous chloride, Cu_2Cl_2, is dissolved by sodium hyposulphite with the formation of double-salts, the nature of which will be fully considered in the next chapter.

§ 224. *Solubility of Lead Sulphate.*

Lead forms a sodium hyposulphite double-salt of the formula $2Na_2S_2O_3, PbS_2O_3$, without water. The solubility of lead sulphate in sodium hyposulphite is characteristic on account of its being, principally, a function of the concentration of the solution. If the same quantity of sodium hyposulphite is dissolved in varying quantities of water, the solubility of the lead sulphate increases materially with the concentration of the solution. The following figures, deduced from a series of Russell's experiments, will illustrate this.

Concentration of the solution in $Na_2S_2O_3+5aq.$	Amount of $PbSO_4$ dissolved for one part of $Na_2S_2O_3+5aq.$
5 per cent.	0.080 parts.
6.6 "	0.102 "
10 "	0.131 "
20 "	0.194 "
25 "	0.226 "

This peculiarity is very favorable for working on a large scale, where hyposulphite solutions of low concentration are used, and the quantity of lead sulphate dissolved is thus brought to a minimum.

The solubility of lead sulphate increases materially with the temperature of the solution. In exposing 20 gm. $PbSO_4$ to 1000 c.c. of a 5 per cent. sodium hyposulphite solution for one hour, 0.048 parts of $PbSO_4$ per unit of $Na_2S_2O_3+5aq.$ were dissolved at 21° C., while the solubility-coefficient at 49° C. was 0.095.

§ 225. *Solubility of Lead Antimonate.*

The solubility of lead antimonate is slight.

1000 c.c. of a 2 per cent. sodium hyposulphite solution dissolved:

At 20° C., 0.147 gm. $PbSb_2O_6$.
" 75° C., 0.306 gm. "

§ 226. *Solubility of Calcium Sulphate.*

If calcium sulphate is dissolved in a sodium hyposulphite solution, sodium sulphate and a double-salt of sodium and calcium hyposulphite are formed.

The solubility of calcium sulphate or gypsum per unit of sodium hyposulphite is, principally, a function of the concentration of the solution. Dilute solutions are relatively better solvents than concentrated ones. The following figures are deduced from a series of Russell's experiments. The solubility of calcium sulphate is decreased by elevating the temperature of the sodium hyposulphite solution, as is the case in dissolving gypsum in water. [One part of $CaSO_4+2aq.$ requires 400 parts of cold water and 460 parts of boiling water for its solution.]

TABLE No. 226.

Showing Solubility of Calcium Sulphate in Sodium Hyposulphite Solutions.

Concentration of the Solution in $Na_2S_2O_3+5aq.$	One part $CaSO_4+2aq.$ required solution:	Amount of $CaSO_4+2aq.$ dissolved for one part $Na_2S_2O_3+5aq.$
5 per cent.	101 parts.	0.198 parts.
6.6 "	84.8 "	0.181 "
10 "	66.2 "	0.151 "
20 "	33.8 "	0.148 "

The experiments were made at about 18° C.

§ 227. *Solubility of Caustic Lime.*

Caustic lime dissolves in sodium hyposulphite solution, a double-salt of sodium and calcium hyposulphite and free caustic soda being formed. Its solubility, however, is not much greater than that in water, diminishing with increase of temperature.

The following quantities of CaO were dissolved by 1000 c.c. of a solution containing 2 per cent. $Na_2S_2O_3+5aq.$:

At 22° C., 1,975 gm. CaO.
" 75° C., 1,295 " "
" 100° C., 1,160 " "

§ 228. *Solubility of Sodium Antimonate.*

Potassium antimonate, added to a solution of sodium hyposulphite, produces the same characteristic precipitate of $Na_2Sb_2O_6$ $+7aq.$ that is obtained with other sodium-salts. The precipitate appears even in very dilute solutions; hence, it must be assumed that the solubility of this salt in sodium hyposulphite is not greater than in water.

1000 c.c. boiling water dissolve 2.86 gm. $Na_2Sb_2O_6+7aq.$

Cold water dissolves less.

CHAPTER III.

THE REACTIONS OF EXTRA-SOLUTIONS.

§ 301. *Cuprous Hyposulphite Double-Salts.*

If solutions of alkaline hyposulphites and of cupric salts are mixed, hyposulphite double-salts, containing the alkali and cuprous oxide, are formed. If the solutions are not too dilute, these double-salts separate in the form of canary-yellow precipitates.

The latter, under certain conditions, turn black in consequence of the formation of cuprous sulphide. By treating them with boiling water, the cuprous hyposulphite is completely decomposed to sulphide. Dilute acids do not decompose these double-salts at ordinary temperature, but do so very quickly upon heating. Some of these double-salts are very little soluble in water, but dissolve readily in aqueous hyposulphite, forming new salts with higher equivalents of the alkaline hyposulphites.

The latter solutions can be heated to 85° C. before Cu_2S commences to separate.

The potassium-salt has, according to Rammelsberg, the formula $K_2S_2O_3$, $Cu_2S_2O_3 + 2aq$. Lenz obtained a sodium double-salt of the formula $2Na_2S_2O_3$, $3Cu_2S_2O_3 + 5aq$. Its formation is expressed by the equation:

(No. 1.) $$11Na_2S_2O_3 + 6CuSO_4 = 2Na_2S_2O_3,\ 3Cu_2S_2O_3 + 6Na_2SO_4 + 3Na_2S_4O_6,$$

leaving out the water-equivalents. It is a reaction similar to the one which takes place if sodium hyposulphite is treated with iodine, viz:

$$2Na_2S_2O_3 + 2I = 2NaI + Na_2S_4O_6.$$

§ 302. *The Standard Extra-Solution.*

The salt $2Na_2S_2O_3$, $3Cu_2S_2O_3 + 5aq.$, which in future will be designated as the salt of Lenz, is very little soluble in water; 1 part requires 352 parts of water for solution at ordinary temperature. In aqueous sodium hyposulphite it dissolves freely, as the figures below demonstrate:

100 c.c.	of a	5	per cent.	solution	dissolve	12.28	gm.
"	"	7½	"	"	"	17.46	"
"	"	10	"	"	"	22.54	"

This gives 2.46 gm., 2.32 gm., and 2.25 gm., of $2N_2S_2O_3$, $3Cu_2S_2O_3+5aq.$ per gm. of $Na_2S_2O_3+5aq.$ respectively.

In calculating the atomic weights, I find that it takes nearly 2 equivalents of sodium hyposulphite to dissolve 1 equivalent of the salt of Lenz. Hence, a double-salt of the formula $4Na_2S_2O_3$, $3Cu_2S_2O_3+x$ aq. must be assumed to exist in the solution.

The same salt would be formed by satisfying the equation:

(No. 2.) $13Na_2S_2O_3+6CuSO_4=4Na_2S_2O_3, 3Cu_2S_2O_3+6Na_2SO_4+3Na_2S_4O_6,$

namely, by mixing solutions containing for 1 part $CuSO_4+5aq.$, $2\frac{15}{100}$ parts $Na_2S_2O_3+5aq.$ A solution of this composition is called a *Standard Extra-Solution.* In actual lixiviation the proportions of the chemicals for this solution are generally so regulated that for each $2\frac{1}{4}$ parts $Na_2S_2O_3+5aq.$, 1 part $CuSO_4+5aq.$ is added. In using these proportions of the reagents, precipitation of the salt of Lenz is prevented.

If the salt of Lenz is dissolved in an excess of sodium hyposulphite, alcohol precipitates from the solution a double-salt of the formula:

$$3Na_2S_2O_3, Cu_2S_2O_3+2aq.$$

All the following reactions, which I have carefully investigated, refer to extra-solutions of moderate concentration, as used in lixiviation on a large scale. Extra-solutions of standard composition, and those containing a much larger quantity of sodium hyposulphite in proportion to copper sulphate, differ so materially in their properties that I shall consider them separately.

I. REACTIONS OF EXTRA-SOLUTIONS WITH CAUSTIC SODA AND SODIUM CARBONATE.

A. EXTRA-SOLUTION OF STANDARD COMPOSITION, CONTAINING FOR 1 PART $CuSO_4+5aq.$, $2\frac{1}{4}$ PARTS $Na_2S_2O_3+5aq.$

§ 303. *Reactions with Caustic Soda.*

Caustic soda precipitates, at ordinary temperature, cuprous hydroxide, CuHO. The precipitate appears, after some time, even in very dilute solutions. The separation of copper is, however, by no means complete. Upon addition of a sufficient amount of sodium hyposulphite, the precipitate is again dissolved. If the cuprous hydroxide is exposed to the air, it is converted into cupric hydroxide, $Cu(HO)_2$, which is insoluble in sodium hyposulphite.

§ 304. *Reactions with Sodium Carbonate.*

Sodium carbonate does not produce, at ordinary temperature, a precipitate. Upon heating the solution above 30° C., a precipitate of cuprous carbonate, Cu_2CO_3, appears, having the same color as the cuprous hydroxide. It is again dissolved upon addition of sodium hyposulphite.

If to a sodium hyposulphite solution either caustic soda or sodium carbonate are added, and then copper sulphate, the characteristic blue precipitates of either cupric hydroxide, $Cu(HO)_2$, or of basic cupric carbonate, $CuCO_3$, $Cu(HO)_2$, appear.

B. THE EXTRA-SOLUTION CONTAINS FOR 1 PART $CuSO_4+5aq.$, $4\frac{1}{2}$ PARTS $Na_2S_2O_3+5aq.$

§ 305. *Reactions with Caustic Soda.*

Caustic soda does not produce a precipitate at ordinary temperature. The latter appears, however, upon heating the solution to about 50° C. If the extra-solution contains more than $4\frac{1}{2}$ parts $Na_2S_2O_3+5aq.$ for 1 part $CuSO_4+5aq.$, a higher temperature is required to separate cuprous hydroxide, but the reaction always takes place before the boiling-point has been reached.

§ 306. *Reactions with Sodium Carbonate.*

Sodium carbonate does not produce a precipitate of cuprous carbonate even if the solution is heated to boiling.

II. DECOMPOSITION OF EXTRA-SOLUTIONS BY HEATING.

§ 307. *A. Extra-Solutions of Standard Composition.*

An extra-solution without free acid commences to decompose if heated above 85° C. A higher temperature rapidly increases decomposition, and, upon boiling, the copper separates completely as Cu_2S, free sulphuric acid being formed under evolution of sulphurous acid. Solutions with a slight amount of free acid commence to turn dark at 65° C.; and the more free acid is present, the lower is the temperature at which decomposition begins.

§ 308. *B. Extra-Solutions containing for* 1 *part* $CuSO_4+5aq.$, $4\frac{1}{2}$ *parts* $Na_2S_2O_3+5aq.$

Extra-solutions of this composition, if free from acid, can be heated to boiling before they turn dark. Even then decomposition is very slow. I have stated, in § 206, that in heating ordinary sodium hyposulphite solution with metallic copper, cuprous hyposulphite remains in solution even after continued boiling. This is explained by the facts stated above.

If the extra-solution contains free sulphuric acid, Cu_2S is separated at a much lower temperature, the degree of the latter depending upon the quantity of free acid present. In all cases the energy of the reaction, after it has once commenced, is increased by the formation of more free acid.

III. DECOMPOSITION OF EXTRA-SOLUTIONS AT ORDINARY TEMPERATURE.

Freshly made extra-solutions always show an acid reaction even if the reagents used were perfectly pure. The commercial bluestone generally encloses acid mother-solution; hence, extra-solution prepared from it may contain free acid.

All extra-solutions undergo decomposition in contact with the atmosphere on account of the great affinity of cuprous salts for oxygen.

A. EXTRA-SOLUTIONS OF STANDARD COMPOSITION.

§ 309. *Solutions without free Acid.*

Solutions without free acid become turbid after mixing. Very soon, a precipitate of cupric hydroxide appears which becomes very copious after 24 hours. The reaction can be explained as follows:

$$2Cu_2S_2O_3+3O+3H_2O=CuS_4O_6+3Cu(HO)_2.$$

The cupric tetrathionate, "in statu nascendi," decomposes sodium hyposulphite:

$$2CuS_4O_6 + 3Na_2S_2O_3 = Cu_2S_2O_3 + 3Na_2S_4O_6.$$

These reactions go on until, finally, nearly all the copper is precipitated as hydroxide, and most of the sodium hyposulphite is converted into tetrathionate:

$$Cu_2S_2O_3 + Na_2S_2O_3 + 2O + 2H_2O = 2Cu(HO)_2 + Na_2S_4O_6.$$

§ 310. *Acid Solutions.*

If a few drops of dilute sulphuric acid are added to the solution, so as to give it a decided acid reaction, it becomes perfectly clear, and remains so for over 24 hours. It would be a mistake, however, to assume that decomposition does not go on. The same reactions take place as pointed out above, with this difference, that deterioration in cuprous hyposulphite is prevented so long as free acid is present. The oxygen absorbed by the cuprous salt is all transferred to the sodium hyposulphite. After sufficiently long exposure to the atmosphere, the free acid becomes neutralized by $Cu(HO)_2$, and the same reactions now go on as in a neutral solution. By this time, the solution has so deteriorated in sodium hyposulphite that a precipitate of the canary-yellow salt of Lenz sometimes appears, besides cupric hydroxide. The decomposition of an acid solution is slow as compared with that of a neutral one. It may be considered as quite permanent for practical purposes, at least so far as cuprous hyposulphite is concerned. A solution with too much free acid is liable to undergo the same alteration, at ordinary temperature, as described in § 307, with separation of Cu_2S.

§ 311. *Solutions to which Sodium Carbonate has been added.*

If sodium carbonate is added to the solution, decomposition is so rapid that the filtrate, under access of air, becomes again turbid as soon as it leaves the filter. Precipitation of cupric carbonate continues until the sodium carbonate is exhausted. In this case the solution suffers only deterioration in cuprous hyposulphite, but not in sodium hyposulphite. The effect is the same as if a cupric salt is added to a hyposulphite solution containing sodium carbonate. Cupric carbonate is formed before a reaction between the cupric and hyposulphite salts can take place. In adding sodium carbonate to a standard extra-solution, a precipitate of cupric carbonate appears so soon that a careless observer might be led to assume that the solution contained a cupric salt, which is not possible.

B. EXTRA-SOLUTIONS CONTAINING FOR 1 PART $CuSO_4+5aq.$, $4\frac{1}{2}$ PARTS $Na_2S_2O_3+5aq.$

These solutions are clear and colorless after mixing.

§ 312. *Solutions without free Acid.*

Solutions without free acid remain clear for many days without depositing cupric hydroxide. That a deterioration in sodium hyposulphite does take place, after several days' exposure, becomes evident upon heating the solution, when it decomposes rapidly, as described in § 307. A freshly made solution, it will be remembered, decomposes very slowly upon boiling. Finally, these solutions deposit Cu_2S, and the more concentrated they are the sooner commences this reaction.

§ 313. *Acid Solutions.*

Solutions to which sulphuric acid has been added soon turn yellow. If the quantity of acid is very slight, the color of the solution does not change for many days, deterioration going on in sodium hyposulphite only. In the presence of more acid, decomposition, as described in § 307, sets in sooner or later, and, if once started, the copper is rapidly precipitated as Cu_2S.

§ 314. *Solutions to which Sodium Carbonate has been added.*

Solutions to which sodium carbonate has been added, either remain clear or become slightly turbid, but, in the latter case, remain clear after filtering. In the decomposition which these solutions undergo, their concentration has some influence. A solution of $2\frac{1}{2}$ per cent. concentration in sodium hyposulphite and $\frac{1}{2}$ per cent. in copper sulphate, does not change, apparently, after many days' exposure. Its deterioration in sodium hyposulphite is, however, proven by the fact that it deposits cuprous oxide upon heating, which is not the case with a fresh solution.

Solutions of 4½ per cent. concentration in sodium hyposulphite and ½ per cent. in copper sulphate, act differently. After 24 hours they assume the color of rich port-wine, indicating the presence of Cu_2S. Indeed, upon boiling, a trace of Cu_2S separates and the solution becomes again colorless. It is difficult to explain why this decomposition should take place in an alkaline solution at ordinary temperature. This decomposition, however, proceeds very slowly.

Upon boiling the solution, after many days' exposure, it becomes turbid, and a more copious precipitate of Cu_2S is formed.

§ 315. *Solutions to which Caustic Soda has been added.*

Solutions to which caustic soda has been added show exactly the same phenomena as described in § 311, with this difference, that cupric hydroxide is precipitated in place of cupric carbonate. Deterioration in cuprous hyposulphite goes on until the caustic soda is exhausted, while sodium hyposulphite remains intact. In solutions with greater excess in sodium hyposulphite the effect of the caustic soda is materially weakened.

§ 316. *The Difference in Reactions Explained.*

It may be asked: what is the reason that extra-solutions of standard composition, and those containing twice the amount of sodium hyposulphite, differ so materially in their reactions? We have to deal with two entirely different double-salts. The one has the formula $4Na_2S_2O_3, 3Cu_2S_2O_3$; the other, $8Na_2S_2O_3, 3Cu_2S_2O_3$. The former is slightly yellow; the latter colorless. The latter salt is also the most permanent of the two, but for that reason it acts with less energy on silver-compounds.

§ 317. *Conclusions of Practical Importance.*

The facts elucidated above are of great practical value. They demonstrate:

1st. That a standard extra-solution is most permanent, at ordinary temperature, if it contains a slight quantity of free sulphuric acid. Such solutions can be safely heated to 50° C.

2d. A standard extra-solution should never be prepared from an alkaline or caustic stock-solution (containing sodium carbonate or caustic soda). Not only is thereby caused a waste in copper sulphate, copper being precipitated as cupric hydroxide or carbonate, but the solution becomes neutral, and, in consequence, liable to speedy deterioration.

3d. An extra-solution containing twice the quantity of sodium hyposulphite, required for standard composition, is most permanent if neutral, free acid producing, at ordinary temperature, decomposition with formation of Cu_2S. Such solutions, without free acid, are especially recommended for application at higher temperatures than 50° C.

4th. If ores containing caustic lime are treated, extra-solution of standard composition deteriorates so quickly that its use becomes unprofitable. Hence, the application of solutions low in copper, but high in sodium hyposulphite, is most advantageous to counteract the deleterious influence of the caustic lime.

5th. It is evident that extra-solution cannot be kept in stock, but must be prepared shortly before application.

§ 318. *Investigation on the Formation of Sulphates in the Decomposition of Extra-Solutions.*

In elucidating the decomposition of extra-solutions by atmospheric influences, the formation of tetrathionates occurs so frequently that the question may be asked, whether sulphates are not formed as well. The latter is, in my opinion, only the case where a decomposition with production of Cu_2S takes place. To prove the correctness of this argument, the following experiment was made. Extra-solution, free from sodium sulphate, was prepared by dissolving the pure salt of Lenz in sodium hyposulphite. $BaCl_2$ precipitates from a moderately concentrated solution of this composition BaS_2O_3, which, however, readily dissolves upon addition of water. The same is the case if $BaCl_2$ is added to a tetrathionate solution. In both of these solutions, containing barium, the slightest quantity of Na_2SO_4 produces a precipitate of $BaSO_4$, proving that this reaction is reliable. The extra-solution was now exposed for 48 hours to the oxidizing influence of the atmosphere, after which time not a trace of sulphate could be detected.

§ 319. *Silver Tetrathionate.*

Another subject needed investigation, namely, the solubility of silver chloride in sodium tetrathionate. To test this, concentrated solutions of copper sulphate and sodium hyposulphite were mixed in proportion to form the salt of Lenz. After separation of the latter, the filtrate, containing Na_2SO_4, $Na_2S_4O_6$ and a slight quantity of the salt of Lenz, was digested with AgCl. The latter was rapidly converted into a heavy, crystalline salt of $Ag_2S_4O_6$. This salt is of so little stability that it decomposes to Ag_2S at very moderate heat. Sodium hyposulphite dissolves it readily. The filtrate from the silver tetrathionate was made acid, and heated to boiling. A slight quantity of Cu_2S and Ag_2S separated. Their solution in nitric acid gave with hydrochloric acid so small a precipitate of silver chloride that its weight was not determined. This proves conclusively that sodium tetrathionate is not a solvent for silver, and that all tetrathionate formed in the preparation and decomposition of extra-solution is impoverishing the solvent energy of a lixiviation-solution.

§ 320. *The Effect of Extra-Solutions upon Silver-Compounds and Silver-Minerals, Compared with Amalgamation.*

The solutions of the cuprous hyposulphite double-salts are peculiar on account of the energetic decomposing action they exert upon most silver-combinations and silver-minerals. That upon silver sulphide has been especially investigated. If the silver sulphide has been freshly precipitated, the reaction is practically instantaneous. In all cases an argentic hyposulphite double-salt is formed, which goes into solution, and copper sulphide is precipitated in case the silver was combined with sulphur.

The most energetic reactions are produced by an extra-solution of standard composition, an increase in sodium hyposulphite as well as in copper sulphate diminishing the effect.

We have here an interesting parallel with the action of cuprous chloride on silver sulphide, and various silver-minerals, upon which the process of raw amalgamation is, principally, based. But there is this difference, that cuprous hyposulphite acts very energetically, even in dilute solutions, while the cuprous chloride must be used more concentrated to decompose silver-minerals, whereby the expense of the process is increased, producing, at the same time, very often, bullion of low fineness in silver.

§ 321. *Extra-Solution with Cuprous Chloride.*

From cuprous chloride, extra-solution is formed according to the reaction :

$$7Na_2S_2O_3 + 3Cu_2Cl_2 = 4Na_2S_2O_3,\ 3Cu_2S_2O_3 + 6NaCl.$$

without loss in sodium hyposulphite. The reaction has a practical bearing in the treatment of ores containing copper. In the chloridizing-roasting of such ores, more or less cuprous chloride is formed, the amount increasing the lower the temperature at which the roasting was conducted.

Base-metal leaching with cold water does not dissolve all of the cuprous chloride in the charge. In the subsequent lixiviation for silver, the hyposulphite solution dissolves the cuprous chloride, thus forming a weak extra-solution. Although the effect of this extra-solution cannot be compared with one of standard composition, it nevertheless has a beneficial effect upon the extraction of the silver. This is one of the reasons why cupriferous silver-ores are more amenable to ordinary lixiviation than those without copper.

This subject will be further discussed in § 1401.

Solubility of Silver, Gold, Lead, Silver Sulphide, and Gold Sulphide in Extra-Solution.

§ 322. *Solubility of Metallic Silver.*

Metallic silver is much easier dissolved by standard extra-solution than by ordinary solution. As is the case with the latter, concentration of the extra-solution does not materially affect the result. Elevation of temperature facilitates the solution of the silver. From a number of Russell's experiments it appears that the best effect, under equal conditions, obtained with extra-solution on metallic silver, is about nine times as great as with ordinary solution, if both are used cold; and about three and a half times as great, if the solutions are heated to 50° C.

Upon what chemical reaction is the greater solubility of metallic silver in extra-solution based as compared with ordinary solution? Evidently, copper cannot be replaced by silver, because the latter is precipitated by the former from a hyposulphite solution. The copper in the extra-solution acts only a secondary part in facilitating the oxidation of the silver. All cuprous salts have a strong tendency to absorb oxygen from the atmosphere, and to act as carriers of this element for oxidation of other substances. That a cuprous hyposulphite solution possesses this property in a high degree has already been fully demonstrated. The reaction goes on indefinitely as long as reducing and oxidizing agents are present simultaneously. It is also evident that concentration of the solution, within certain limits, cannot influence the result. The absorption of oxygen by the solution is, principally, a function of time, temperature, and surface exposure. Standard extra-solutions absorb oxygen much quicker than those containing an excess of sodium hyposulphite. For this reason they are more energetic solvents for metallic silver.

§ 323. *Solubility of Metallic Gold.*

A series of experiments was made, as in § 204, with substantially the same results, showing the dissolving energy of extra-solution upon metallic gold, to be equal, not superior, to that of ordinary solution. (Russell.) The slight affinity of gold for oxygen, which is not sufficiently stimulated by the presence of a cuprous salt, explains this fact. A different result may be expected in treating gold not in the form of gold-leaf or cement-gold, but in finely divided condition, as obtained by reduction from gold sulphide. (§ 326.)

§ 324. *Solubility of Metallic Lead.*

Metallic lead precipitates copper from standard extra-solutions quite rapidly, even at ordinary temperature, especially if free acid is present. In solutions with an excess of sodium hyposulphite, the reaction is decidedly slower, and, in the absence of free acid, it is only observed after some time.

325. *Solubility of Silver Sulphide.*

Ag_2S is readily and rapidly decomposed, and the silver dissolved, if treated with extra-solution, especially if the latter is warm. Standard extra-solutions were prepared by saturating sodium hyposulphite solutions of various concentrations with the salt of Lenz.

The effect of such solutions on Ag_2S was as follows, namely:

Saturated solution of the salt of Lenz in aqueous sodium hyposulphite of	Silver dissolved per gm. of the salt of Lenz.
5 per cent. concentration.	0.137 gm.
7½ " "	0.115 "
10 " "	0.113 "

A solution of the salt of Lenz in water dissolved only 0.022 gm. silver per gm. of this salt.

The equivalent of the salt of Lenz is 1120; that of silver, 108. If all the copper had been replaced by silver, 0.578 gm. Ag should have gone into solution per gm. of the salt of Lenz. The actual result is less than ¼ of the above figure. It has to be considered, however, that in order to effect a complete decomposition, there is not sufficient sodium hyposulphite in a standard extra-solution to form the easily soluble double-salt, $2Na_2S_2O_3$, $Ag_2S_2O_3$. On this basis a solution of 0.385 gm. Ag might be expected. Numerous experiments made by Russell have established the fact that an increased quantity of sodium hyposulphite, in place of augmenting, materially diminishes the solvent energy of extra-solution. Hence, deficiency in sodium hyposulphite, necessary for the formation of the easily soluble argentic double-salt, cannot be the cause that the quantity of silver dissolved is so small. Evidently, the reaction is checked after a certain amount of silver has entered the solution. This is confirmed in working on a large scale. It is always necessary to use an excess of extra-solution to dissolve silver-compounds that cannot be extracted by ordinary lixiviation. A more logical explanation of these observations can be given as follows. It has been demonstrated that the permanency of the cuprous hyposulphite double-salts increases the more equivalents of sodium hyposulphite they contain in proportion to cuprous hyposulphite. The least permanent extra-solution is one of standard composition; hence, it acts most energetically as a solvent for silver. In dissolving silver, the standard extra-solution is more and more impoverished in copper, and its permanency is increased until, finally, its solvent power for silver becomes extremely weak.

§ 326. *Solubility of Gold Sulphide.*

Au_2S_3 is readily decomposed by extra-solution. In the following experiments, 0.500 gm. gold were dissolved in aqua regia, and precipitated with H_2S. The Au_2S_3 was treated with 1000 c. c. of a standard extra-solution containing 1½ per cent. $CuSO_4+5aq$. After one hour, 0.460 gm. gold had gone into solution; after an hour and a half, another sample showed nearly all the gold, namely, 0.482 gm., dissolved. Also in this case copper sulphide is precipitated.

In another series of experiments, 1.333 gm. gold were, in each case, converted into Au_2S_3, and treated with extra-solutions of varying concentration, and at different temperatures. The results were not uniform. The solutions acted best at a moderate temperature, and heating them seemed to be of no benefit. Evidently, upon heating the solutions, Au_2S_3 is decomposed to metallic gold, and the latter is less amenable to extra-solution than the sulphide.

§ 327. *Solubility of Silver Arsenate and Antimonate.*

The effect of extra-solution on these silver-salts does not differ from that of ordinary solution. (Russell.)

THE EFFECT OF EXTRA-SOLUTION ON SILVER-ORES.

We now arrive at the most important part of the experiments,—so far as their direct practical value is concerned, *i. e.*, the effect of extra-solution in extracting silver from ores. Regarding the assay-office lixiviation-tests, it will suffice to state, for the present, (this subject will be fully treated in a separate chapter) that small quantities of ore, ¼ or ½ A. T., are treated with an excess of both ordinary and extra-solution, and the tailings are assayed for silver.

§ 328. *The Effect of Extra-Solution on Raw Ores.*

Most interesting are the results which Russell obtained in lixiviating silver-ores without previous roasting.

TABLE No. 328. a.

NAME OF MINE.	Process by which ores are worked at present.	Silver per ton of ore. Ounces.	Silver extracted by Lixiviation with:		Difference. Per cent.	Character of Ore.
			Ordinary Solution. Per cent.	Extra-Solution. Per cent.		
Lexington, Montana	Roasting and amalgamation.	54.0	26.0	68.5	42.5	Native silver, silver-bearing pyrites of iron, zinc-blende, galena, pyrites of copper.
Manhattan Company, Nevada	Roasting and amalgamation.	164.8	8.3	57.2	48.9	Group of antimonial and arsenical sulphurets of silver, fahl-ore, zinc-blende, galena, pyrites of iron and copper.
Ontario, Utah	Roasting and amalgamation.	90.8	7.0	31.9	24.9	Fahl-ore, native silver, zinc-blende, not much pyrites of iron and copper, galena.
Custer, Idaho	Roasting and amalgamation.	30.8	27.8	72.8	45.0	Principally stephanite, pyrites of iron, etc.
Ramshorn, Idaho	Smelting.	89.6	34.4	50.5	16.1	Principally iron carbonate, some lead minerals, pyrites of copper, copper-silver glance.
Mines near Durango, Mexico	Patio process.	58.0	2.0	39.8	37.8	Completely oxidized, no sulphurets visible. Below water-level the ore contains galena. The oxidized ore, most likely, contains lead sulphate and carbonate.
		14.8	16.9	51.4	34.5	
		32.8	8.0	65.3	57.3	
		19.2	34.9	70.4	35.5	
Tombstone, Arizona	Raw amalg'mat'n. Battery sands.	52.8	70.5	81.1	10.6	Oxidized or free-milling ores, with more or less silver chloride and lead sulphate and carbonate.
	Sands from tailings beds.	9.2	60.9	70.0	9.1	
	Slimes from tailing beds.	17.2	81.4	81.4	none.	
Sierra Grande, New Mexico	Tailings from raw amalgamation, concentrated	67.6	61.1	70.2	9.1	Oxidized or free-milling ores, with silver chloride and lead sulphate and carbonate.
Sombretillo, Sonora, Mexico	?	86.4	75.0	84.0	9.0	Gangue composed of about 30 p. c. quartz and 70 p. c. carbonate of lime and magnesia. Silver, most likely, present as chloride.
Ore from dump of Chrysolite, Leadville, Colorado		8.0	24.0	40.0	16.0	Silver combined with lead sulphate and carb. Also silver chloride.

The ores in the tabulated statement on page 41 belong mostly to the so-called "base ores"; yet the percentage of silver extracted is, in many cases, remarkably high.

From this table it appears that, of the sulphuret-ores, those are principally affected by extra-solution which carry the silver in the form of native silver, silver sulphuret, and the group of antimonial and arsenical sulphurets, like pyrargyrite, stephanite, and polybasite. Being especially familiar with the ores from the Lexington, Manhattan Company's, and Ontario mines, a comparison of results will be interesting. The Lexington ore is the basest of the three, and any one seeing this ore in bulk would not suppose that raw lixiviation could extract 68.5 per cent. of its silver. Some of its silver occurs native, and the principal portion of it is combined with iron pyrites as silver sulphide. Then comes the ore from the Manhattan mines. Native silver here is rare. The principal silver-bearing minerals are ruby-silver, stephanite, and polybasite. Ontario ore shows the lowest result. Native silver occurs, but to much less extent than at the Lexington. About 88 per cent. of its silver is found as fahl-ore, or its products by decomposition, and this mineral seems to be more refractory in contact with extra-solution.

These tests were made by Russell, in 1883, with ore-samples taken at that time from the above named mines.

The battery-sample from Tombstone is of interest, because it shows that Russell's process will extract from such ores a higher percentage than raw amalgamation. In the annual report—covering the time when these samples were taken—of the Tombstone Mill and Mining Company, the yield of silver by raw amalgamation is given at 76 per cent. of the assay-value of the ore.

Below is another series of lixiviation-tests with raw ores from various mines.

TABLE No. 328. b.

Name of Mine.	Value of Ore. Ounces p. t.	Silver extracted by Lixiviation with: Extra-solution. Per cent.	Ordinary solution. Per cent.	Difference. Per cent.
Grand Central, Arizona,	43.2	93.7	86.6	7.1
Horn Silver, Utah,	184.0	90.7	81.6	9.1
Price River, Utah,	25.0	87.2	77.6	9.6
Silver Reef, Utah,	45.0	77.8	50.7	27.1

The ore-samples in the preceding tables were all pulverized so as to pass either through a No. 30 or a No. 40 screen. The following figures show the influence of finer crushing.

Sample of Oxidized Ore from the 200-foot Level of the Ontario Mine. Value, 48.8 ounces Silver per ton.

	Pulverized so as to pass. No. screen.	Lixiviation-test with: Ordinary solution. Per cent.	Extra-solution. Per cent.
	20	50.0	60.0
	40	52.0	65.6
	60	53.3	67.0
	80	55.0	75.5
Difference between No. 20 and No. 80 screens,		5.0	15.5

Sample of Ore from the Yedras Mine, Mexico. Value, 33.6 ounces Silver per ton.

	Pulverized so as to pass. No. screen.	Lixiviation-test with: Ordinary solution. Per cent.	Extra-solution. Per cent.
	40	45.3	50.6
	60	45.3	58.4
	80	45.3	64.3
Difference between No. 40 and No. 80 screens.		none.	13.7

From these results it appears that fine crushing increases the extraction of the silver. The same is necessary in raw amalgamation with chemicals, where the ore is finely ground in the pan after a more or less coarse crushing in the battery. In both cases the particles of the silver-minerals, on account of their density, are not penetrated by the solution, and the reaction takes place near the surface only. It is quite different with roasted ores. Crushing through a No 20 screen is, in most cases, sufficient for good roasting, whereby the particles of ore are left in a porous condition. It is well known that the expense of crushing increases materially with fineness, and that the latter augments the mechanical difficulties in treating raw ores by lixiviation. Hence, it may be, in such cases, more profitable to crush the ore as coarse as possible, and first roast it before extracting the silver by lixiviation. With well appointed mechanical furnaces, of large capacity, the expense of roasting, so far as labor and fuel are concerned, is slight. The cost of the salt is, in chloridizing-roasting, the principal item of expense.

I refer here to the fact that lixiviation of raw ores has been practiced at the Old Telegraph mine, Utah. The ore contained silver chloride and lead-minerals. The latter were concentrated after lixiviation. In attempting to concentrate the ore at once, the silver chloride was lost. There is no doubt that the extra-solution would have extracted a much higher percentage of the silver. This system may be used to advantage in all cases where the character of the ore is such that a large percentage of the silver is lost by direct concentration, and where this portion of the silver is in a form soluble in extra-solution.

§ 329. *The Effect of Extra-Solution on Ores that have been Oxidized by Roasting.*

In view of the energetic reactions of the extra-solution, I considered it of interest to carry out the idea of lixiviating ores after they had been subjected to an oxidizing-roasting. These experiments were, at first, made on a small scale, *i. e.*, by roasting samples in a muffle.

Table No. 329. a. contains the results of Russell's experiments. In each case a sample of 500 gm. was roasted.

In reviewing this table it is, in the first place, evident that ores become less fit for lixiviation the longer they have been roasted. The two Ontario samples, roasted at different temperatures, show the injurious effect of high heat. If we compare the lixiviation-tests in the above table with those in the previous paragraph, made with raw ores and extra-solution, we find that in all cases, excepting the Ontario ore, the roasting proved to be decidedly detrimental.

The curious fact, that, after oxidizing-roasting of silver-ores, the percentage of silver extracted by hyposulphite solutions decreases the more the roasting is prolonged, led to the question: What will be the result if the roasting is done in the shortest possible time, namely, in a Stetefeldt furnace? This experiment was carried out at the Ontario mill. After cleaning out the chloridized ore from the hoppers of the shaft and flue, the feeding of the salt was suspended for two hours. Samples of raw ore, taken every half hour from the feeder, showed the pulp to contain only a fraction of 1 per cent. of salt, the latter being derived from pulp lodged in the troughs of the conveyor-screws. The roasting

TABLE No. 329. a.

Ore from	Temperature.	Time. hours.	Per cent. of silver extracted with:		Per cent. of silver lost in roasting.
			Ordinary solution.	Extra-solution.	
Ontario Mine.	Cherry-red.	½	31.1	55.8	2.2
		1	28.6	40.0	8.8
		1½	19.3	28.9	10.4
		2	17.0	26.9	11.0
		2½	17.3	23.0	13.4
Ontario Mine.	Dark-red.	½	44.2	64.5	0.7
		1	35.1	44.5	7.0
		1½	31.0	34.8	7.1
		2	24.6	26.4	8.7
		2½	23.8	26.8	8.3
Ramshorn Mine.	Dark-red.	½	13.4	17.4	2.3
		1	7.7	11.6	5.0
		1½	7.0	7.6	5.3
		2	6.6	7.7	5.4
		2½	3.9	5.5	5.8
Manhattan Mine.	Dark-red.	½	10.0	22.2	4.8
		1	5.0	18.1	6.5
		1½	8.0	13.0	12.4
		2	2.6	----	13.0
		2½	1.0	2.0	14.0
Lexington Mine.	Dark-red.	½	17.6	32.5	3.2
		1	6.2	12.0	3.7
		1½	6.6	6.6	4.0
		2	1.2	4.7	4.3
		2½	2.0	5.5	4.6

was conducted at a very high temperature and with an abundant supply of air for oxidation. Samples of roasted ore were taken from the top of the ore accumulated in the shaft and flue, and lixiviated.

Sample from :	Silver extracted with : Ordinary solution. Per cent.	Extra-solution. Per cent.
Shaft	19.0	58.6
"	28.8	55.6
Flue	14.4	45.5
"	39.9	67.3

At the expiration of two hours, the ore was discharged from the shaft and flue, allowed to remain in piles on the cooling-floor for twelve hours, and was then carefully sampled. Below are the results:

Silver extracted by treatment with:	Ore from: Shaft. Per cent.	Ore from: Flue. Per cent.
Water	17.5	4.9
Water and H_2SO_4	16.3	5.2
Ordinary solution	86.0	62.4
Extra-solution	89.7	69.2

These results were so interesting and important, and so contrary to accepted metallurgical principles, that it was considered expedient to repeat the experiments.

The following five experiments were made with Ontario ore which carried an unusually large percentage of zinc-blende. As in the previous case, a fraction of 1 per cent. of salt (from three-tenths to six-tenths) remained in the ore. If we call the amount of fire generally used in chloridizing-roasting at the Ontario mill "normal," this condition was only maintained in experiment No. 4. In all other cases extra fire was used, especially in Nos. 1, 2, and 3. The final results, that is, after the ore has remained on the cooling-floor, indicate that this extra fire is beneficial. In experiment No. 3, the admission of air was materially reduced, and the temperature was highest. The initial result in this case is the lowest of all, but the increase in per cent. of silver extracted, after the ore had remained on the cooling-floor, is by far the highest, and the final result the best. When the ore in experiment No. 4 was discharged, it was not red-hot. Although the initial result is much better than in No. 3, the gain by remaining on the cooling-floor is less than one-half of No. 3.

The first question which presents itself in reviewing Table No. 329. b. is: In what condition does the silver exist in the roasted ore? The silver soluble in water is, undoubtedly, present as silver sulphate. From the fact that a large percentage of silver is extracted by ordinary solution, it follows that the principal part of the silver must be in combinations equally soluble in ordinary and extra-solution. As such we have found the arsenates and antimonates of silver. Nearly all the silver in Ontario ore occurs as fahl-ore, and from it these combinations are derived. Some silver, it is fair to presume, is present in metallic form, and this is easily soluble in extra-solution. Silver sulphide as such does not occur in the ore, and supposing it to be present, is so easily oxidized that it cannot exist after roasting.

TABLE OF EXPERIMENTS. No. 329. b.

Samples taken directly from shaft and flue.		Experiment.				
		No. 1.	No. 2.	No. 3.	No. 4.	No. 5.
Value of raw ore, oz. per ton		74.8	83.2	82.4	102.8	79.5
Value of roasted ore, shaft, oz.		66.	68.	62.	92.	72.
Value of roasted ore, flue, oz.		94.	84.	84.	108.	91.
Per cent. of salts soluble in water	shaft	6.5	3.0	3.5	3.2	4.2
	flue	2.0	1.5	1.5	5.0	2.1
Per cent. of salts soluble in sodium hyposulphite	shaft	14.0	7.5	9.5	10.7	9.3
	flue	9.0	5.0	6.5	7.0	6.0
Per cent. of silver extracted by leaching with water	shaft	5.2	5.3	5.8	4.8	5.6
	flue	12.0	----	----	4.0	----
Per cent. of silver extracted by ordinary solution	shaft	30.9	20.9	31.5	23.0	27.5
	flue	29.0	26.9	15.0	25.2	18.8
Per cent. of silver extracted by extra-solution	shaft	66.5	60.6	50.0	59.6	66.6
	flue	61.6	70.9	45.5	66.3	67.8

Samples taken after the ore had remained 16 hrs. on cooling-floor.	Composition of charge, about, parts.				
	3 shaft, 2 flue.	3 shaft, 2 flue.	Shaft only.	4 shaft, 1 flue.	Shaft only.
Per ct. of salts soluble in water	8.5	5.0	6.7	5.5	5.5
Per ct. of salts sol. in sod. hypo.	13.2	9.5	12.7	11.0	10.0
Per ct. silver extracted by water	2.0	----	12.6	6.8	10.0
Per cent. of silver, ordinary sol.	74.2	64.8	83.4	35.3	73.7
Per cent. of silver, extra-solution	84.0	80.0	85.8	77.6	81.9
Increase of per cent. in silver extracted by extra-solution as compared with the samples taken directly from shaft and flue, considering the composition of charge.	19.5	15.3	35.8	16.7	15.2
Per cent. of silver extracted by lixiviating 2 tons.	85.0	72.1	85.7	74.4	80.6

The solubility-tests of the roasted ore show a marked difference between shaft and flue. After the roasted ore had remained red-hot for a considerable time on the cooling-floor, the quantity of salts soluble in water and sodium hyposulphite was materially increased. The sulphates of copper, zinc, and manganese reacted most energetically on the silver, increasing the quantity of silver sulphate, antimonate and arsenate, thus converting it into combinations soluble in hyposulphite solutions.

From the facts recorded about the influence of time in muffle-roasting, it appears that a momentary roasting is essential to the success of the process.

In other words, the longer the ore has been exposed to the influence of external heat in the process of oxidation, the less soluble become the silver-compounds in hyposulphite solutions. It is doubtful whether instantaneous roasting produces silver-combinations differing from those formed in muffle-roasting, and the difference in solubility is caused thereby; or is the result in muffle-roasting due merely to the long exposure of the ore to heat? As to the latter cause, analogous phenomena are not wanting in chemistry. I allude to the peculiarity of some metallic oxides becoming less soluble in acids after they have been exposed to a high temperature for a long time.

I cannot end this discussion without pointing out the practical importance of these experiments. If it is possible to extract silver by lixiviation without using salt in roasting, we not only save the expense of the salt, but also the cost of drying and crushing it, and of the plant needed for these purposes. For localities where salt is very costly, and the ores are of low grade, a decrease in the yield of silver, of several per cent., would still leave a balance in favor of oxidizing-roasting, especially when silver is so low in price as at present.

THE EFFECT OF EXTRA-SOLUTION ON ORES THAT HAVE BEEN SUBJECTED TO A CHLORIDIZING-ROASTING.

We have to make here a distinction between roasted ores free from caustic lime, and those containing it in a perceptible degree. The former are first lixiviated with ordinary solution, and then the extra-solution is applied. If roasted ores containing caustic lime are treated in the same way, the extra-solution sometimes fails to work properly; but if it is applied at once, the cuprous hyposulphite neutralizes and counteracts the damaging effect of the caustic lime, and leaves the silver in a soluble form. Exceptions to this rule will be noted later.

§ 330. *A. Roasted Ores free from Caustic Lime.*

Lixiviation-Tests at the Ontario mill.—Russell made lixiviation-tests of the Ontario mill-samples both with ordinary and extra-solution, and compared the results with those obtained by amalgamation.

The Ontario ore is roasted with salt in two Stetefeldt furnaces. After discharging the ore from the furnaces, it is piled up on the cooling-floor and left for 12 hours or longer, undisturbed. It is then spread out, cooled with a spray of water, and loaded into cars for charging the pans. From these cars the samples were taken to which reference is made below. The percentage of silver, extracted from the roasted ore by amalgamation, is calculated from the silver remaining in the tailings by comparing their value with the value of the roasted ore after leaching out its soluble salts with water.

Averages of lixiviation-tests, and percentage of silver extracted by amalgamation at the Ontario mill.

	Lixiviation-test with:				Silver extracted by
Samples taken from:	Ordinary solution.		Extra-solution.		amalgamation.
	Per cent.		Per cent.		Per cent.
March, '82, to April, '83	88.1		92.6		90.4
Differences		4.5		2.2	

The effect of extra-solution on Ontario ore, if not well chloridized.—Much more pronounced becomes the difference between the dissolving energy of the two solutions if ore is lixiviated which does not show a high chlorination by the ordinary method. Experience has established the fact that Ontario ore, and all ores with a large percentage of sulphurets, especially zinc-blende, require considerable time to complete the chlorination of the silver. This is effected by leaving the ore on the cooling-floor in a red-hot state for 12 hours or longer. There are ores which do not necessarily require this treatment, but in all cases the effect is more or less beneficial. If samples of Ontario ore are taken immediately after a charge has been drawn from the furnace, the percentage of the silver chloridized is very low, especially if the ore has been roasted at a moderate temperature. The dust seems to make an exception to this rule, but, in fact, it does not, if we consider that it remains in the dust-chambers for a long time before it is discharged.

TABLE No. 330. a.

Samples of Ontario ore lixiviated immediately after discharging from the Stetefeldt Furnace.

From where discharged:	Lixiviation-test with: Ordinary Solution. Per cent.	Lixiviation-test with: Extra-solution. Per cent.	Differences: Per cent.
Shaft	60.7	86.1	25.4
Return-flue	61.8	84.2	22.4
Dust-chambers	90.7	93.3	2.6
Shaft	65.2	93.2	28.0
Return-flue	60.7	91.8	31.1
Dust-chambers	80.0	91.4	11.4

TABLE No. 330. b.

Lixiviation-tests made with Roasted Ores from Various Mills.

Furnace Used.	Name of Mill.	Ordinary Solution. Per cent.	Extra Solution. Per cent.	Difference. Per cent.
Howell	Alice, Montana	73.7	89.2	15.5
"	" "	86.0	92.0	6.0
Brückner	Custer, Idaho	83.3	91.5	3.2
Stetefeldt	Lexington, Montana	92.4	93.6	1.2
Stetefeldt	Manhattan, Austin	94.1	94.6	0.5
Howell	Black Warrior, Arizona	94.0	95.2	1.2
Brückner	Bertrand, Nevada, Mt. Cory Ore.	89.5	95.1	5.6

Whenever the chlorination of the silver has been carried to a very high percentage, it cannot be expected that the results of lixiviation with the two solutions should differ very much. Hence, it will be necessary to investigate in every special case how much profit can be derived from the use of the extra-solution.

§ 331. *B. Roasted Ores containing Caustic Lime.*

In lixiviating roasted ores containing caustic lime, the difference in extraction of silver between ordinary and extra-solution is much more pronounced. As a prominent example I cite some results from the Yedras mill, Mexico. The ore contained from 50 to 60 ounces silver, per ton, and was roasted in reverberatory-furnaces.

TABLE No. 331. a.

Lixiviation-tests made with Roasted Yedras Ore.

Per cent. Silver extracted.		
Ordinary Solution.	Extra-Solution.	Difference. Per cent.
77.9	90.0	12.1
66.8	91.0	24.2
48.6	78.5	29.9

Table No. 331. b. gives some results from the Aspen mill, Colorado. The ore, assaying about 25 ounces silver, per ton, contained lime and magnesia, and was roasted in a Stetefeldt furnace.

TABLE No. 331. b.

Lixiviation-tests made with Roasted Aspen Ore.

Per cent. Silver extracted.		
Ordinary Solution.	Extra-Solution.	Difference. Per cent.
67.6	92.8	25.2
72.2	93.0	20.8
75.8	90.6	14.8

§ 332. *Effect of Extra-Solution on Tailings.*

Tailings from ores that have been amalgamated, either raw or after roasting, contain a considerable portion of their silver in combinations soluble in hyposulphite solutions. Hence, it may be profitable to lixiviate such tailings, unless they are in the form of very fine slimes, and offer insurmountable difficulties to filtration. The same applies to tailings from the ordinary lixiviation-process from which the extra-solution will extract more silver. Russell tested a great number of tailings-samples from the Ontario mill, finding that from 20 to 50 per cent. of the silver could be lixiviated. I obtained similar results with tailings from the Lexington mill. Below is more evidence of the same character.

TABLE No. 332.

Lixiviation-tests made with Tailings from various Mills.

Name of Mill.	Process.	Silver extracted by: Ordinary Solution. Per cent.	Extra-Solution. Per cent.	Difference. Per cent.
Sierra Grande, New Mex.	Raw amalgamation	61.1	70.2	9.1
Black Warrior, Arizona	Roasting and amalgamation	57.5	62.5	5.0
Bertrand, Nevada	Roasting and ord. lixiviation	7.2	59.8	52.6
Raymond and Ely, Nev.	Raw amalgamation	30.4	42.4	12.0

CHAPTER IV.

THE EXTRACTION OF GOLD.

The problem of extracting from gold-bearing silver-ores a high percentage of both precious metals, by the application of one humid process, has not yet been solved. If such ores are suitable to raw amalgamation in pans, the yield in gold is often quite satisfactory, while that in silver remains low. Chloridizing-roasting improves results so far as silver is concerned, but often diminishes the yield in gold. Although the lixiviation-process with hyposulphite solution is not adapted to working gold-ores proper, it enters into successful competition with amalgamation in beneficiating gold-bearing silver-ores, after they have been roasted with salt. Russell made a number of tests with ore from the Lexington mill. The ore carried from 42 to 50 ounces silver and from 0.5 to 0.9 ounce gold, per ton, and had been chloridized in a Stetefeldt furnace. Lixiviation extracted from 60 to 70 per cent. of the gold. The statistics of the Lexington mill, at that time, recorded a yield in gold of only 50 to 53 per cent. by amalgamation. R. D. Clark worked roasted ore from the Alice mine, at the Bertrand mill, by lixiviation, and also reported a better extraction of the gold than that obtained at the Alice mill by amalgamation. According to W. G. Lamb, the gold-value of Ontario and Daly ore was, in 1891, about $1.00 per ton. Of this, amalgamation extracted at the Ontario mill 47 per cent., and lixiviation extracted at the Marsac mill (from Daly ore) 71 per cent. Ontario and Daly ore are extracted from the same vein. In a series of comparative tests made with Daly ore, in 1893–94, the average extraction of gold by lixiviation was 71 per cent., and that by amalgamation 63 per cent.

I found that the percentage of gold extracted by amalgamation decreases with the increase of temperature at which the roasting is conducted, and with the time consumed in roasting, or the time during which the charge is kept red-hot after being withdrawn from the furnace. Unfortunately, in most cases, amalgamation requires, for a high extraction of the silver, roasting at a highly elevated temperature, and keeping the charge red-hot on the cooling-floor for a long time.

Such conditions are not always needed if the ore is lixiviated by the extra-solution of the Russell process. Hence, a quick roasting in a Stetefeldt furnace may leave the ore in a condition most favorable for a high extraction of both silver and gold.

CHAPTER V.

CHEMISTRY OF THE WASH-WATER.*

§ 501. *The Composition of Roasted Ores.*

Ores that have been subjected to a chloridizing-roasting contain a great number of salts soluble either in water or in brine. The salts soluble in water are, principally, sodium sulphate and chloride, a large portion of the latter remaining undecomposed in roasting. Besides, there may be the sulphates of manganese, zinc, copper, iron, aluminum and magnesium; the chlorides of the same metals, and of calcium, barium and strontium, the latter being of rare occurrence. The chlorides of barium and strontium, however, would be immediately decomposed on solution, and insoluble sulphates of these metals precipitated. Calcium chloride would also be changed to sulphate. Finally, there may be sodium arsenate. Salts not so easily soluble in water, as those mentioned above, are: cuprous chloride, lead chloride, calcium sulphate, sodium antimonate, and caustic lime. Lead chloride, if dissolved, cannot remain as such, but will be precipitated as lead sulphate. Silver chloride, lead sulphate and antimonate, are almost insoluble in water, but more or less soluble in concentrated wash-water brine. In the same way the solubility of cuprous chloride, calcium sulphate, and caustic lime is increased by the presence of sodium chloride, and other salts, in water. If caustic lime exists in the ore in sufficient quantity, the soluble salts of the base metals are decomposed with precipitation of hydroxides. If soluble base-metal salts predominate, the caustic lime may be completely neutralized.

Since all roasted ores have to be leached with water, prior to the application of the hyposulphite solutions, a considerable percentage of the silver in the ore may be taken up by the so-called first wash-water.

---

* The solubility of various salts, given in this chapter, is mostly taken from Biedermann's Chemiker Kalender.

§ 502. *Necessity of Leaching with Water.*

Leaching with water is necessary:

1st. So as not to overload the lixiviation-solution with foreign salts.

2nd. Because the sulphides, precipitated from the lixiviation-solution, would become very base if the salts of the base metals were not removed, as much as possible, prior to the extraction of the silver.

It is of importance to know the solubility of the principal salts which come into play in base-metal leaching; hence, this subject will be considered in detail.

A. SALTS EASILY SOLUBLE IN WATER.

§ 503. *Solubility of* Na_2SO_4.

1000 c. c. water dissolve:

At 18° C., 532.5 gm. Na_2SO_4.
" 30° C., 503.7 " "
" 50° C., 468.2 " "
" 100° C., 426.5 " "

§ 504. *Solubility of* $NaCl$.

1000 c. c. water dissolve:

At 0° C., 355.2 gm. $NaCl$.
" 14° C., 358.7 "
" 25° C., 361.3 "
" 50° C., 369.8 "
" 100° C., 396.1 "

§ 505. *Solubility of* $CaCl_2$.

1000 c. c. water dissolve:

At 10° C., 633.5 gm. $CaCl_2$.
" 40° C., 1204.8 "
" 100° C., 1550.0 "

§ 506. *The Sulphates and Chlorides of Manganese, Zinc, Copper* ($CuCl_2$), *Iron, Aluminum, and Magnesium.*

These are all easily soluble in cold and in hot water. Sodium arsenate is also soluble. The above named salts exist in roasted ores in comparatively small quantities so that a detailed account of their solubilities is not of much importance.

B. SALTS NOT FREELY SOLUBLE IN WATER.

§ 507. *Solubility of* Cu_2Cl_2.

1000 c. c. water dissolve:

At 14° C., 0.9 gm. Cu_2Cl_2.
" 100° C., 1.35 "

§ 508. *Solubility of* $PbCl_2$.

1000 c. c. water dissolve:

At 12.5° C., 7.4 gm. $PbCl_2$.
" 100° C., 50.0 "

§ 509. *Solubility of* $CaSo_4+2aq$.

1000 c. c. water dissolve:

At 0° C., 1.90 gm. $CaSO_4+2aq$.
" 18° C., 2.05 "
" 36° C., 2.14 "
" 53° C., 2.11 "
" 99° C., 1.75 "

§ 510. *Solubility of CaO.*

The solubility of CaO in water decreases with increase in temperature. Solubility-coefficients, given by different authorities, do not agree very closely. According to A. Lamy's investigations, the solubility is influenced by the method of preparing CaO. For CaO obtained by calcining pure marble, Lamy gives the following figures:

1000 c. c. of a saturated solution contain:

At 0° C., 1.381 gm. CaO.
" 15° C., 1.299 "
" 30° C., 1.162 "
" 60° C., 0.868 "
" 100° C., 0.576 "

According to Dalton:

1 gm. CaO, at 15° C., requires 778 c. c. water, and 1 gm. CaO, at 100° C., requires 1270 c. c. water for solution.

Bineau gives the following solubilities:

1 gm. CaO, at 18° C., requires 780 c. c.
" " 100° C., " 1500 c. c.

water for solution.

§ 511. *Solubility of Sodium Antimonate.*

1000 c. c. of boiling water dissolve 2.86 gm. $2NaSbO_3+7aq.$, or 1.73 gm. $NaSbO_3$. The solubility in cold water is much less.

C. Salts not easily soluble in Water, but more soluble in Brine.

§ 512. *Solubility of Cu_2Cl_2.*

1000 c. c. concentrated brine dissolve:

At 40° C., 80.0 gm. Cu_2Cl_2.
" 90° C., 160.0 "

1000 c. c. brine, with 15 per cent. NaCl, dissolve:

At 14° C., 35.0 gm. Cu_2Cl_2.
" 84° C., 60.0 "
" 90° C., 100.0 "

1000 c. c. brine, with 5 per cent. NaCl, dissolve:

At 40° C., 14.0 gm. Cu_2Cl_2.
" 90° C., 26.0 "

These figures are, according to T. Sterry Hunt, approximate only. Both on cooling and diluting these solutions with cold water, Cu_2Cl_2 is precipitated.

§ 513. *Solubility of $PbCl_2$.*

1000 c. c. concentrated brine dissolve:

At about 15° C., 10.8 gm. $PbCl_2$.
At boiling point, 53.7 "

§ 514. *Solubility of $CaSO_4+2aq.$*

According to Anthon, 1000 c. c. concentrated brine dissolve, at ordinary temperature, 8.2 gm. $CaSO_4+2aq.$; hence for 1000 gm. NaCl present in the solution, 25 gm. $CaSO_4+2aq.$ are dissolved.

An analysis of brine, saturated with gypsum, from Koesen, Germany, shows for 8.52 per cent. NaCl, 0.63 per cent. $CaSo_4+$

2aq. in solution; 1000 c. c. of this brine contain, at ordinary temperature, 6.3 gm. $CaSO_4+2aq.$, or per 1000 gm. NaCl present, 74 gm. $CaSO_4+2aq.$ are in solution. This would indicate that dilute brine is comparatively a better solvent for gypsum than concentrated brine even if we deduct the quantity of gypsum dissolved by water alone. I have found no data as to influence of temperature, but judging by analogy from the solubility of gypsum in water, and in sodium hyposulphite solutions, I assume that its solubility in brine is diminished by increase of temperature.

D. Salts practically insoluble in Water, but soluble in Brine and in solutions of other Chlorides.

In this respect I have found data for silver chloride only.

§ 515. *Solubility of AgCl in Brine.*

According to Frémy and Pelouze, the quantity of AgCl dissolved in brine increases in proportion to the units of NaCl present in the solution, and, for solutions of equal concentration, rises with elevation of temperature as follows:

At 0° C., only traces of AgCl are dissolved.

At 10° C., 1.7 gm. AgCl per 1000 gm. NaCl.
" 18° C., 2.4 " " "
" 100° C., 6.8 " " "

It is not stated whether these chemists employed saturated solutions. Vogel found that 1000 c. c. concentrated brine dissolved, at ordinary temperature, 0.95 gm. AgCl. If Vogel operated at a temperature of about 18° C., his result would give per 1000 gm. NaCl in solution, 3.0 gm. AgCl dissolved, a value greater than that quoted by Frémy and Pelouze. This difference may be accounted for if Vogel's solubility-coefficient refers to a higher temperature than 18° C.

Greatly at variance with the above figures are the observations of H. Hahn, who found that a solution of sodium chloride, containing 25.96 per cent. NaCl, and of 1.2053 specific gravity, dissolved, at 19.6° C., per 1000 c. c., 1.269 gm. AgCl, or 4.05 gm. AgCl per 1000 gm. NaCl. The difference between Hahn and Vogel may be accounted for by difference in temperature; that between Hahn, and Frémy and Pelouze, by the latter not having used concentrated solutions. The solubility of AgCl in brine is evidently a complex function. It increases for solutions of equal concentration with a rise in temperature, and also increases at equal temperatures for solutions holding equal amounts of NaCl, with the concentration of the solution. The latter view is supported by observations of Fresenius.

§ 516. *Solubility of AgCl in solutions of various Chlorides.*

In this respect, the observations of different chemists show, apparently, still greater discrepancies than those noted above.

Concentrated solutions of $CaCl_2$, $MgCl_2$, and $BaCl_2$, prepared by Vogel at ordinary temperature, and by H. Hahn at 24.5° C., but used at 30.6° C., dissolved per 1000 c. c.:

Solution.	Vogel.	Hahn.
$CaCl_2$	0.930 gm.	8.34 gm. AgCl.
$MgCl_2$	0.710 "	7.09 "
$BaCl_2$	0.143 "	0·74 "

To unravel the mystery as to $CaCl_2$, E. Jackson, according to Percy, prepared a concentrated solution of $CaCl_2$ at ordinary temperature, added a surplus of AgCl, and found per 1000 c. c. of the solution dissolved:

After 2 days	0.714 gm. AgCl.
" 5 "	0.765 "
" 10 "	1.000 "
" 21 "	1.506 "
" 60 "	1.894 "

These results agree much better with those of Vogel than with Hahn's. I believe that the apparent discrepancy between Hahn's results, and those of Vogel and Jackson, can be easily explained. Neither Vogel nor Jackson say at what temperature their solutions were saturated with $CaCl_2$, or used for dissolving AgCl. Now, the solubility of $CaCl_2$, $BaCl_2$, and $MgCl_2$, increases rapidly with a rise in temperature, and the solvent power of these solutions for AgCl also increases both with their concentration and elevation of temperature. Hence, without knowing the temperature at which Vogel and Jackson operated, their results are not comparable with those of Hahn, whose higher solubility-coefficients must be due both to a higher temperature and concentration of the solutions.

According to Hahn, the chlorides of manganese, iron, copper, and zinc are also good solvents for AgCl. The solutions were saturated at 24.5° C.

1000 c. c.	$MnCl_2$	solution dissolved	2.956	gm. AgCl.
"	$FeCl_2$	"	2.393	"
"	$CuCl_2$	"	0.833	"
"	$ZnCl_2$	"	0.215	"

Fresenius, speaking of the solubility of silver chloride in hot, concentrated solutions of the chlorides of sodium, potassium, and calcium, says: "On sufficient dilution with cold water, the dissolved silver chloride separates so completely that the filtrate is not colored by sulphuretted hydrogen."*

The following solubility-tests were made by the author with brine as it may be obtained in applying the first wash-water to roasted ores.

It is assumed, in one case, that in roasting a base ore, free from lime, $\frac{1}{3}$ of the salt remained undecomposed, while $\frac{2}{3}$ were converted into sodium sulphate; and that the first concentrated portion of the wash-water held per 1000 c. c., 80 gm. NaCl, and 195 gm. Na_2SO_4. In another case, a solution of the same character was saturated with caustic lime and gypsum, or with gypsum alone, to represent a wash-water brine resulting from a roasted base ore containing lime.

R. SALTS INSOLUBLE IN WATER, BUT SOLUBLE IN WASH-WATER BRINE.

§ 517. *Solubility of Silver Chloride.*

1000 c. c. wash-water brine dissolve:

At 20° C.	0.036 gm. AgCl.
" 75° C.	0.172 "

1000 c. c. wash-water brine, saturated with gypsum, dissolve:

At 20° C.	0.039 gm. AgCl.
" 75° C.	0.166 "

1000 c. c. wash-water brine, saturated with gypsum and caustic lime, dissolve:

At 20° C.	0.036 gm. AgCl.
" 75° C.	0.166 "

---

* Whether this is also the case with brine containing sodium sulphate and salts of copper, iron, zinc, etc., Fresenius does not state.

§ 518. *Solubility of Silver Arsenate and Antimonate.*

The solubility of silver arsenate and antimonate in brine is the same as that of silver chloride, so far as the amount of silver dissolved is concerned. If precipitates of the former salts are added to brine, they are decomposed with formation of silver chloride. In a solution of silver chloride in brine, potassium arsenate does not produce a precipitate; and upon diluting such a solution with cold water, silver chloride separates, and not silver arsenate.

In treating roasted ores containing silver arsenate and antimonate, the conversion of these salts into silver chloride in contact with brine is evidently very incomplete in consequence of their fused condition. I draw this conclusion from the fact that in pan-amalgamation silver arsenate and antimonate remain in the tailings.

§ 519. *Solubility of Lead Sulphate.*

1000 c. c. wash-water brine dissolve:

At 20° C. 0.092 gm. $PbSO_4$.
" 75° C. 0.192 " "

1000 c. c. wash-water brine, saturated with gypsum, dissolve:

At 20° C. 0.073 gm. $PbSO_4$.
" 75° C. 0.440 " "

1000 c. c. wash-water brine, saturated with gypsum and caustic lime, dissolve:

At 20° C. 0.393 gm. $PbSO_4$.
" 75° C. 0.476 " "

§ 520. *Solubility of Lead Antimonate.*

The solubility of lead antimonate in brine, so far as the quantity of lead dissolved is concerned, is the same as that of lead sulphate, the latter salt going into solution after decomposition of the lead antimonate.

The results of solubilities in the last paragraphs should not be considered as a correct measure of the quantities of silver and lead that may be taken up by the wash-water. Their importance lies principally in this, that they show a great difference in the effect of leaching with cold and with hot water. In operating on a large scale, the conditions are different, and the character of the solutions is much more complex on account of the presence of other salts, besides sodium chloride and sulphate.

§ 521. *Conclusions.*

With a knowledge of these facts, the question whether it is more advantageous to use cold or hot water in base-metal leaching is easily decided. The salts of group A.: Na_2SO_4, NaCl, $CaCl_2$, and other sulphates and chlorides, are so easily soluble, both in cold and hot water, that not much benefit is derived by the use of the latter. In group B., $CaSO_4+2aq.$ and CaO are more soluble in cold than in hot water; hence, preference should be given to the former. Cu_2Cl_2 is less soluble in cold than in hot water; but leaving Cu_2Cl_2 in the charge is beneficial, because it forms extra-solution in subsequent lixiviation with hyposulphite solutions. The same argument holds good for Cu_2Cl_2 as considered in group C. Finally, AgCl being very much less soluble in cold than in hot brine, cold water is best used on that account also. With exception of lead sulphate, the evidence is altogether in favor of cold water; but even a hot wash-water brine does not dissolve much lead sulphate. If roasted ore contains an excess of caustic lime, the caustic wash-water brine cannot hold in solution iron and manganese; but it may contain copper,* zinc, lead, and aluminum, besides antimony and arsenic. Caustic lime does not prevent or materially influence the solution of silver chloride in brine. If the wash-water brine contains calcium chloride, its solvent power for silver chloride will be materially increased. Calcium chloride, however, can only be present in the solution if it exists in excess of sodium sulphate, since, by mixing the two salts, gypsum is formed, which is partly dissolved by brine. A solution of gypsum in brine is, as we have seen, not a better solvent for silver chloride than brine itself.

The first, concentrated brine, which leaves the ore-tanks in base-metal leaching, if diluted with sufficient cold water, becomes turbid, and AgCl, Cu_2Cl_2, and $PbSO_4$, besides basic salts of the base metals, are precipitated.

---

* Russell observed that wash-water giving a caustic reaction may contain copper. This would indicate that copper is not completely precipitated by caustics from a solution of cuprous chloride in brine, analogous to the reactions with sodium hyposulphite solutions, as shown in § 303 and § 305.

§ 522. *Effect of the first Wash-Water on the Chlorination of the Ore, or the so-called "going back" of Chlorinations.*

If base ores containing zinc-blende (and this mineral is rarely absent) are treated, after roasting, by lixiviation, a sample of the washed ore, *i. e.*, after the first wash-water has been applied in the ore-tank, may show a much lower chlorination, or extraction by ordinary solution, than the original charge-sample. This is especially the case if the ore has been subjected to a quick roasting in a Stetefeldt or Howell furnace; it is less marked where the ore has been roasted perfectly dead in a reverberatory or Brückner furnace.* W. S. Morse has given to this phenomenon the name "going back" of chlorination, which I herewith introduce into metallurgical literature. He has also made, at the Aspen mill, Colorado, valuable observations regarding the cause of the "going back."

In most mills separate chlorination-tests of the washed ore are not made; hence the "going back" does not appear in the assay-office reports. It becomes evident, however, from the fact that the ordinary solution extracts much less silver in the mill than might be expected from the assay-office tests of the charge-samples.

Should the ore contain zinc-blende, and at the same time so much carbonate of lime (or magnesia) that a part of the latter is not converted into sulphate by roasting, but remains as caustic lime, then the "going back" of chlorinations is aggravated by two causes.

Mr. Morse has kept, at the Aspen mill, complete statistics on the "going back" of chlorinations, and these will be given before discussing the theory of the problem. The mill treated, in 1892, over 30,000 tons of ore which contained on an average 10.99 per cent. CaO; 4.24 MgO; 2.85 Zn; and 27.9 ounces silver, per ton.

The average "going back" of chlorinations was as follows :†

---

* About chlorinations "going back" on ore roasted in Brückner furnaces, see § 1903.

† In figuring the percentage of chlorination of the washed ore, the value of the tailings, after treatment with ordinary or extra-solution, is compared with the calculated value of the charge-sample of roasted ore, not with the value of the washed ore.

	Per cent. Silver Extracted.	
Assay-Office Results.	Ordinary Sol.	Extra Sol.
Charge-sample	78.92	89.78
Washed Ore	64.32	88.57
“ Gone back ”	14.60	1.21
Mill Results.	58.77	86.74
Behind Assay-Office on Charge-sample	20.15	3.04
“ “ “ Washed Ore.	5.55	1.82

The mill result, ordinary solution, refers to the apparent extraction by the first wash-water and the first ordinary solution applied to the charge. It could hardly be expected that this extraction should come up to the chlorination of the washed ore. The mill result by extra-solution refers to the final apparent extraction.

Now, Mr. Morse gives the extremes of these averages as follows:

Beginning of March, the raw ore contained 18.3 per cent. CaO and MgO, and 4.5 Zn, of which 3.74 per cent. was present as ZnS.

	Per cent. Silver Extracted.	
Assay-Office Results.	Ordinary Sol.	Extra-Sol.
Charge-sample	71.16	86.36
Washed Ore	34.00	83.10
“ Gone back ”	37.16	3.26
Mill Results.	18.70	80.80
Behind Assay-Office on Charge-sample	52.46	5.56
“ “ “ Washed Ore.	15.30	2.30

In October, the raw ore contained: 11.2 per cent. CaO and MgO, and 1.8 Zn, of which 1.21 per cent. was present as ZnS.

	Per cent. Silver Extracted.	
Assay-Office Results.	Ordinary Sol.	Extra-Sol.
Charge-sample	82.33	89.93
Washed Ore	79.15	90.15
“ Gone back ”	3.18	Increased 0.22
Mill Results.	77.33	90.05
Behind Assay-Office on Charge-sample	5.00	Increased 0.12
“ “ “ Washed Ore.	1.82	0.10

Mr. Morse says: The “going back” of chlorinations is produced by a portion of the zinc-blende remaining unroasted in the ore. As soon as the charge is saturated by the first wash-water, the following reaction takes place: $2\,AgCl + ZnS = Ag_2S + ZnCl_2$

In support of this he cites the experiments of Malaguti and Durocher, a full account of which is found in Percy's Metallurgy, Silver and Gold, Part I. 98.

There is no doubt that this explanation is correct. Of all base metal sulphurets, zinc-blende is the most difficult to oxidize, and requires a higher temperature and more time for a dead roast than any other sulphuret. Even prolonged heap-roasting on the cooling-floor of a Stetefeldt furnace does not seem to effect materially that portion of the zinc-blende which has escaped oxidation in dropping down the shaft.

That ZnS in roasted ore is the principal and in many cases the only cause of chlorinations "going back" becomes evident from Table No. 522. Mr. Morse had ZnS determined in average samples of roasted ore (charge-samples) from 20 lots, and compared the results with the percentage of chlorinations "gone back" of the washed ore in the mill.

TABLE No. 522.

SHOWING THE RELATION BETWEEN ZnS IN ROASTED ORE AND THE "GOING BACK" OF CHLORINATIONS OF WASHED ORE.

Chlorinations "gone back" on Washed Ore. Per cent.	Zn present as ZnS in Roasted Ore. Per cent.	Tons of Ore from which. Average Sample was taken.	Chlorinations "gone back" on Washed Ore. Per cent.	Zn present as ZnS in Roasted Ore. Per cent.	Tons of Ore from which Average Sample was taken.
37.16	3.16	1319	9.66	1.64	2602
32.41	2.86	2090	9.18	1.14	2380
24.93	2.59	1231	7.86	1.36	1910
22.30	2.00	1309	4.53	1.05	541
19.58	2.07	2435	4.00	0.89	410
18.62	2.18	3255	3.42	0.63	480
16.14	2.04	1276	3.32	1.00	2687
15.00	1.87	961	3.18	0.68	2860
13.70	0.82	2123	0.51	1.15	2656
12.58	1.25	1065	0.24	0.97	2033

It will be noted that in the beginning of Table No. 522 the percentage of chlorinations "gone back" is in proportion to the percentage of ZnS in the roasted ore. At the end of the table this relation is more or less variable, and we must assume that some other causes came into play which either counteracted the reaction between ZnS and AgCl, or else intensified the "going back" by reaction not due to ZnS. In conclusion, Mr. Morse says: When I suspected the cause of our difficulties and reduced the zinc*, we got splendid results, as shown below.

Month.	ZnS in Raw Ore. Per cent.	Apparent Extraction in Mill. Per cent.	Actual Clean-up in Sulphides, calculated on Silver in Roasted Ore. Per cent.
July	0.82	88.80	95.1
August	1.00	90.18	99.0
September	1.15	90.25	99.5
October	0.68	90.05	95.8

Those who are working lime-ores, requiring the addition of sulphurets in roasting, will profit by Mr. Morse's costly experience and use pyritic ores containing copper in place of zinc.

It may be asked: Why is a comparatively high chlorination of the charge-sample obtained in the laboratory? The reason is that the reaction takes considerable time. A laboratory chlorination-test is quickly made, but the washing of a deep charge in an ore-tank takes 24 hours and longer. At the Aspen mill samples of roasted ore were washed in the laboratory with water for 15 minutes, and gave an average chlorination of 74.3 per cent. Samples of the same ore, left in contact with water for 12 hours, gave an average chlorination of only 54.9 per cent., a "going back" of 19.4 per cent. It is evident that the reaction cannot be instantaneous. The silver chloride in roasted ore is in a fused condition and not everywhere in contact with undecomposed zinc-blende. Time is also required to effect the solution of the silver chloride in brine.

That the extra-solution should correct the "going back" of the chlorination is not astonishing, because it easily dissolves silver sulphide.

Now, a few words about the effect of caustic lime. It has been shown, in § 218, how much caustics diminish the solvent energy of a hyposulphite solution in extracting silver from roasted ores. In § 219, a theory has been advanced explaining the phenomenon. The same theory holds good for that portion of the "going back" of chlorinations of washed ore which is caused by caustic lime. In this instance, however, the extra-solution does not bring relief and the loss is irreparable. The deleterious effect of the caustic lime is largely neutralized by filling the ore-tank with a weak solution of bluestone into which the ore is dumped.

---

* The zinc was derived, principally, from pyritic ores, imported from Leadville, which were added to the Aspen ores in order to increase the sulphur, and counteract the deleterious effect of caustic lime in roasting. Ores high in zinc were bought on account of the high working-charges received.

The Aspen statistics, given above, seem to indicate that that portion of the "going back" of chlorinations which is not corrected by the extra-solution, is due to the effect of caustic lime. Mr. Morse says: Where caustic lime is present, if not neutralized by blue-stone or acid in the first wash-water, the result is always low top tailings and high bottom tailings, and no amount of extra-solution corrects this.

The high bottom tailings can be accounted for as follows: The first wash-water, in filtering through the charge, becomes more caustic the longer it remains in contact with the charge and the nearer it gets to the bottom of the tank; hence, the deleterious effect is most pronounced at the bottom. Finally, I give the results of experiments made at my request by Frank A. Bird, chemist of the Anchor mine, Park City, Utah. Zinc-blende, as pure as it could be obtained (it contained 6 per cent. of lead and 4.4 ounces of silver, per ton) was finely pulverized in a porcelain mortar. A portion of it was roasted in a muffle for 22 minutes, leaving about 40 per cent. of its sulphur unoxidized. Samples of the raw and partially roasted blende, in quantities of ½ A. T. = 14.58 gm., were then left in contact with 100 c. c. of silver-bearing, concentrated brine and hyposulphite solutions at a temperature of about 21° C. The brine contained, per 100 c. c., 64.1 mgm. of silver as $AgCl$, and the hyposulphite solution, of 2 per cent. concentration in $Na_2S_2O_3 + 5$ aq., contained 100 mgm. of silver per 100 c. c. The time of exposure varied between 6 and 24 hours. In washing the residues from the brine, hot, concentrated brine was used to prevent precipitation of $AgCl$. The residues were then assayed, deducting the silver originally contained in the zinc-blende. The percentage of silver precipitated as Ag_2S refers to the total silver originally contained in 100 c. c. of the solutions.

The Samples Stood for 6 Hours.

With Brine.				With Hypo. Solution.			
Raw Zinc-blende.		Roasted Zinc-blende.		Raw Zinc-blende.		Roasted Zinc-blende.	
Silver Precipitated.				Silver Precipitated.			
mgm.	Per cent.	mgm.	Per cent.	mgm.	Per cent.	mgm.	Per cent.
42.58	66.43	10.62	16.57	38.22	38.22	8.3	8.3
The Samples Stood for 12 Hours.							
38.44	59.97	14.8	23.09	42.8	42.8	9.58	9.58
The Samples Stood for 24 Hours.							
43.2	67.39	14.4	22.46	49.58	49.58	10.08	10.08

It appears from this that the reaction of the silver-bearing brine is much more energetic and quicker than that of the hyposulphite solution, although the latter contained considerably more silver than the former; also that the reaction of the brine is more irregular, depending, most likely, upon slight differences in temperature, etc.

CHAPTER VI.

THE PRECIPITATION OF METALS FROM HYPOSULPHITE SOLUTIONS.*

In the Ziervogel and Augustin processes, metallic silver is precipitated from the solutions by copper. Although base metals also precipitate metallic silver from hyposulphite solutions, this method is not practicable for the following reasons:

In the first place, the precipitation is slow, and it is incomplete if copper is used, because the resulting extra-solution re-dissolves silver. The principal objection is, however, that the stock-solution is weakened in effective hyposulphite salts while, on the contrary, precipitation by sodium or calcium sulphide not only regenerates the stock-solution, but increases its strength by adding hyposulphite contained in the precipitant.

Precipitation of the silver by electrolysis has been suggested, but this is open to still greater objections. These are: The expense of the plant; the slowness of the process; the precipitation of silver sulphide under certain conditions; and the decomposition of hyposulphite salts of silver and other metals.

The metals playing the most important part in precipitation from hyposulphide solutions, resulting from the lixiviation of silver-ores, are silver, copper, lead, and calcium. The quantity of gold that may be present is too small to enter into calculation. Silver and copper are always precipitated as sulphides by sodium or calcium sulphide; lead and calcium may be precipitated as carbonates by Solvay soda; or lead alone as hydroxide by caustic lime. Sodium sulphide can be used either as Na_2S, or as a mixture of Na_2S and Na_2S_2, which we will designate in future as "Russell's Sulphide," or as Na_2S_2 or a higher polysulphide. Calcium sulphide is always used as CaS_5. For precipitating lead, caustic milk of lime is prepared with hyposulphite solution. Solvay soda, which is in the shape of a fine powder, has been used without dissolving it, but this is bad practice; it should be first dissolved in stock-solution.

---

* Trans. Am. Inst., Vol. xx, p. 15.

Calcium may also be precipitated by a sodium sulphide solution, as will be fully discussed later on. Caustic lime and Solvay soda are always used before calcium or sodium sulphide.

In the absence of calcium, the lead carbonate precipitated by Solvay soda can be obtained practically free from silver, that is to say, after washing it with fresh hyposulphite solution. From Ontario ore, for instance, Russell obtained in this way lead carbonate with only two ounces of silver, per ton. If calcium is present, the lead-calcium carbonate contains more or less silver, and may be very rich. According to Wilson and Russell, the silver does not increase in proportion to the calcium precipitated. What influences the richness of the precipitate in silver, outside of calcium, is not known.

At the Marsac mill, for instance, one lot of carbonates was produced containing 8.1 per cent. lead and 392 ounces silver, per ton; and another lot containing 19 per cent. lead and 1269 ounces silver, per ton.

Lead hydroxide, precipitated by caustic lime, is also rich in silver. According to Rueger, the lead-precipitate obtained at the Mount Corry mill, Nevada, contained 60 per cent. lead and 420 ounces silver, per ton, as already stated previously.

The separate precipitation of lead and calcium may be not only economical directly, but also indirectly by the production of silver sulphides of higher grade.

Since the introduction, by Kiss, of CaS_5 as precipitant, the majority of metallurgists have tenaciously adhered to this practice, evidently without examining the subject critically. The question has become more complicated by the recent introduction of methods for separate precipitation of lead, and of the Russell process. It is hardly necessary to add that the problem also includes the question whether a sodium or calcium hyposulphite solution deserves preference in lixiviation.

Although lixiviation is always commenced with sodium hyposulphite, this salt is gradually replaced by calcium hyposulphite if CaS_5 is used as precipitant.

In the following I propose to present in a systematic manner the most important facts that have a practical bearing upon the subject of precipitation.

§ 601. *Preparation of Sodium and Calcium Sulphides in the Mill.*

Both reagents are prepared by boiling either a concentrated lye of caustic soda, or caustic milk of lime with sulphur. In making calcium sulphide, sufficient sulphur must be added to the caustic lime to obtain CaS_4, because lower calcium sulphides are not formed. In fact, the solution contains CaS_4 even if an excess of lime is present, while an excess of sulphur produces CaS_5. From the slight solubility of calcium hydrate in water it follows that this process must require considerable time, and that, at the end of the operation, a solution of only moderate concentration must result. From sodium hydrate, on the contrary, a lye of almost any concentration can be obtained, as well as sodium monosulphide, and all of the polysulphides. Hence, the process must be completed very rapidly, and, if the lower polysulphides are desired, with a minimum consumption of sulphur.

Russell discovered a practical and easy method of preparing sodium sulphide. According to his directions, the contents of a drum of caustic soda are broken into lumps and dissolved with the aid of steam in a minimum of water,* thus forming a very concentrated lye. The operation is conducted in a cast-iron tank 3 feet diameter and 7 feet high. When the lye has reached a temperature of not less than 100° C., sulphur in lumps is gradually added, two-thirds of the weight of the caustic soda. A most violent reaction takes place, the mass foaming considerably. This, however, is caused by disengagement of steam and not of any other gas. After all the sulphur has disappeared, the product is allowed to cool, dissolved by adding hyposulphite stock-solution, and discharged into a storage-tank, where the solution can be diluted still further. In two hours, 700 pounds caustic soda, the contents of one drum, can be easily converted into sodium sulphide. The solution contains a mixture of Na_2S and Na_2S_2.

The quantity of sulphur used for making Russell's sulphide is somewhat empirical. The two-thirds rule may be very convenient for the laborer; but metallurgy has not been invented for his convenience. As will be seen at once, the rule does not at all consider different grades of caustic soda, which may contain from

---

*In case the steam is wet, no water is added.

85 to 95 per cent. NaHO, thus producing sodium sulphide differing materially in composition. If we use sulphur with 95 per cent. S, and caustic soda with 95 per cent. NaHO, the resulting sulphide will be $Na_2S + Na_2S_2$. If, however, the caustic soda contains only 85 per cent. NaHO, the sulphide will have very nearly the composition $Na_2S + 4\ Na_2S_2$.

For making calcium sulphide, Kuestel recommends using for one part of sulphur, 1⅓ parts caustic lime of purest quality, or 3 parts of the latter if it is impure, and 10 parts water. The caustic lime is charged into an iron tank, then water is added, and finally the pulverized sulphur. The latter should be sufficiently fine to pass a No. 30 wire-screen. Steam is admitted through an iron pipe, and the mass is kept boiling for 4 or 6 hours, stirring it frequently. More water, if necessary, must be added to keep the charge sufficiently liquid. At the end of the operation, the solution is allowed to stand undisturbed for 12 hours, when it appears clear above the undissolved residue. It is now drawn off. Its concentration should be at least 6° B. In the residue will be found orange-colored crystals, which are, according to Schoene, $3CaO, CaS_4 + 12aq$. They require about 400 parts of water for their solution. The quantity of these crystals increases, the shorter the time of boiling, and the less sulphur used in proportion to the lime. Hence, it is economy to use an excess of sulphur. In that case the solution contains CaS_5, not CaS_4.

§ 602. *Formation of Hyposulphite Salts.*

The reactions taking place in boiling caustic soda with sulphur are expressed by the equation:

$$(1)\quad 6\ NaHO + (2+2x)S = 2Na_2S_x + Na_2S_2O_3 + 3H_2O.$$

That is to say, for two equivalents of a sodium sulphide, one equivalent of the hyposulphite salt is formed.

For the formation of different sodium polysulphides, the quantities of sulphur consumed would be as follows:

100 c. p. Caustic-Soda require c. p. Sulphur.	100 Caustic Soda, with 90 per cent. NaHO, require Sulphur with 95 per cent. S.	For the formation of
53.3	50.5	Na_2S.
80.0	75.8	Na_2S_2.
106.6	101.0	Na_2S_3.
133.3	126.3	Na_2S_4.
160.0	151.6	Na_2S_5.

In manufacturing sodium sulphide there are no very important side-issues to the reactions expressed by equation No. 1; and, taking into consideration the impurities of commercial caustic soda and sulphur, calculations for practical purposes can be based upon them. There are, however, certain limitations to the equations. It is not true, for instance, when $x = 1$. In using an amount of sulphur only sufficient for the formation of Na_2S, caustic soda remains free, and polysulphides are formed besides Na_2S. It is even difficult to obtain a reagent absolutely free from caustic soda by using enough sulphur to form Na_2S_2. Such a solution is also liable to contain some Na_2S and polysulphides higher than Na_2S_2, a tendency prevailing to form the higher polysulphides. If we replace in equation No. 1 natrium by calcium, the reactions become still more complicated. In the first place, the equation is only true for CaS_4 and CaS_5. It is very probable, however, that some sulfhydrates are also formed, and this seems the case even in preparing sodium sulphide. I draw attention to the fact that H_2S frequently escapes at the end of precipitation, even if the solution had originally an alkaline reaction. This is the case in using both calcium and sodium sulphide as precipitants. The formation of H_2S can be easily explained if we assume the existence of sulfhydrates. The reaction would be as follows: $Ag_2S_2O_3 + CaH_2S_2 = Ag_2S + Ca_2S_2O_3 + H_2S$. Naturally, the H_2S only escapes when precipitation of the metals is nearly completed.

This also explains an observation made by Russell at the Marsac mill.

He says: "The solution running from the ore-tanks to the precipitating-tanks is perfectly neutral, but after precipitation with sodium sulphide it is so acid that blue litmus-paper is instantly reddened by it. At the end of precipitation the odor of H_2S is very perceptible; it even appears during precipitation if stirring is not well done."

This, it seems to me, shows the presence of sulfhydrates. The H_2S formed in the beginning of precipitation is absorbed by the solution, throwing down metals and setting acid free. To return to the calcium sulphide:

The reaction is also disturbed by the formation of oxysulphurets —which are not easily soluble—especially if sulphur is not used in excess to form CaS_5, and by decomposition of calcium hyposulphite at boiling-point. While a sodium hyposulphite solution can be heated to boiling without detriment, that of the calcium-salt decomposes above 60° C. as follows:

$$CaS_2O_3 = CaSO_3 + S.$$

This decomposition is, however, resisted in the presence of CaS_4 and CaS_5, and counteracted by the opposite reaction:

$$CaSO_3 + S = CaS_2O_3,$$

which takes place between 30° and 40° C., that is to say, after cooling and settling of the solution. To what extent these opposing reactions balance each other is not known, and depends, most likely, upon various circumstances.

For these reasons it is not possible to calculate beforehand, even approximately, from the chemicals consumed, the effect of a calcium sulphide solution in precipitation, or to compare it, merely by calculation, with the same effect of a sodium sulphide solution. Only actual mill-statistics make a comparison between the two reagents possible. It can readily be seen, however, that the preparation of CaS_5 must involve a waste of chemicals compared with that of Na_2S_2.

According to equation No. 1, 100 parts of caustic soda, consumed in the manufacture of sodium sulphide, will produce 103.3 parts of $Na_2S_2O_3 + 5$ aq.

Or 100 parts commercial caustic soda, containing 90 per cent. NaHO, would produce 93 parts $Na_2S_2O_3 + 5$ aq. Both sodium and calcium sulphide solutions, if exposed to the air, oxidize with formation of hyposulphite salts, but the latter more rapidly than the former.

In case Na_2S is exposed to the atmosphere, sodium hydrate is formed besides hyposulphite:

(2.) $2Na_2S + H_2O + 4O = Na_2S_2O_3 + 2NaHO$.

The caustic soda is then converted into carbonate by absorbing carbonic acid from the air. Na_2S_2 is completely converted into hyposulphite, while the higher polysulphides are oxidized with precipitation of free sulphur.

O. Hofmann, who used CaS_5 as precipitant at the Silver King mill, Arizona, stated to me that the original stock-solution was used over a year and a half, and that it increased in strength and volume, making it necessary to run a part of it to waste.

In this case a large amount of copper and lead was precipitated with the silver. This is good evidence to show how rapidly CaS_5 oxidizes. What has been said about the more rapid decomposition of calcium sulphide in contact with air, compared with sodium sulphide, holds good regarding the respective hyposulphite salts.

Formerly, lixiviation-solutions were never heated; the new practice finds it profitable to operate with solutions of a temperature as high as 50° C. Would not the loss in calcium hyposulphite be much increased in such a case, when complete decomposition takes place above 60° C? The stubborn advocates of the old practice in lixiviation say: We never needed to buy sodium hyposulphite; our stock-solution increased in strength, and we had to run it finally to waste! But at what expense, gentlemen, in caustic lime and sulphur, did you reach this result, leaving alone your inferior extraction in silver?

§ 603. *Economy of Producing Hyposulphites by Oxidation of Sulphide Solutions.*

The question is of importance, whether it is cheaper to buy and add sodium hyposulphite to a deteriorated stock-solution, or to introduce hyposulphite by allowing the sulphide solution to oxidize, whereby it degenerates in precipitating power, but gains in hyposulphite. This depends upon local prices of chemicals. In most cases it will be cheaper to produce the hyposulphite by oxidation of the sulphide. A calculation under assumed conditions will be of interest. For Na_2S_2 we find:

The formation of 100 pounds $Na_2S_2O_3 + 5aq$ requires:

35.8 pounds caustic soda of 90 per cent., at 5.5 cts.	$1.96
27.2 pounds sulphur of 95 per cent., at 3.5 cts.	0.95
Total cost	$2.91
100 pounds $Na_2S_2O_3+5aq$, at 4 cts.	4.00
Leaving a saving of	$1.09

if this salt is obtained by the oxidation of Na_2S_2 in the mill.

The cost of obtaining calcium hyposulphite from the sulphide solution by oxidation cannot be accurately or even approximately calculated, for reasons already stated. From the fact that CaS_2 must be oxidized in place of Na_2S_2, and that in the preparation of calcium sulphide relatively more chemicals are consumed, it follows that the cost cannot, in most cases, be materially less, although caustic lime is very much cheaper than caustic soda; and it may be even greater.

To use a calcium sulphide solution so highly oxidized that the stock-solution increases in strength and volume, making it necessary to run it to waste, cannot be good economy, because it necessitates an excessive consumption of the precipitant.

Should it be desirable to use an oxidized sodium sulphide solution, containing a large amount of hyposulphite, oxidation is most conveniently and quickly effected by forcing air through a coil of gas-pipe, provided with a great number of small holes, and placed at the bottom of the storage-tank.

From an experiment made at the Aspen mill, blowing air through a solution of the commercial Na_2S crystals, it appears that oxidation, *i. e.*, formation of hyposulphite is very slow. Mr. Arents, on the contrary, found, at the Mt. Cory mill, that a calcium sulphide solution is quickly oxidized in this way, especially if warm. The following experiment was carried out at the Marsac mill with sodium sulphide solution made by boiling caustic soda with sulphur, as described in § 601.

Air was blown for 12 hours through a cold solution of such concentration as used in the mill. The increase in sodium hyposulphite was 65.8 gm. per 1000 cc. The effect would, no doubt, have been greater with a warm solution.

§ 604. *Precipitation of Calcium Carbonate by Sodium Sulphide Solutions.*

I have observed in former paragraphs:

1st. That it is difficult to prepare the lower sodium polysulphides absolutely free from caustic soda.

2d. That if a sodium sulphide solution containing Na_2S is exposed to the atmosphere, $Na_2S_2O_3+2NaHO$ are formed.

3d. That free caustic soda absorbs carbonic acid from the atmosphere and becomes Na_2CO_3.

In these reactions lies the key to the appearance of the mysterious precipitate which Russell mistook for an insoluble calcium sulphide. The light-yellow precipitate appears after nearly all the silver, copper, and lead have been precipitated from a lixiviation-solution containing calcium-salts. I do not deny that the precipitate appears under the circumstances; but to pronounce it a calcium sulphide on account of its apparent color, and without analytical investigation, was a mistake.

As can be readily seen, Russell's sulphide, if for any length of time exposed to the atmosphere, must contain more or less Na_2CO_3. The Na_2CO_3 originally contained in commercial caustic soda—which may vary between 3 and 5 per cent.—also comes into play. Assuming, for instance, that 6 pounds caustic soda were consumed (manufactured into sodium sulphide) per ton of ore, containing 4 per cent. Na_2CO_3, there would come into action 0.24 pounds Na_2CO_3, precipitating 0.22 pounds $CaCO_3$. Assuming that Russell's sulphide had been made of a high-grade caustic soda (with 95 per cent. NaHO), having the composition $Na_2S+Na_2S_2$, and that by long exposure to the air, one-half of the Na_2S became oxidized, how much Na_2CO_3 would a quantity of Russell's sulphide contain, produced from 6 pounds caustic soda? Exactly 1.28 pounds, precipitating 1.20 pounds $CaCO_3$. Adding the sodium carbonate originally present in the caustic soda, the total amount would be 1.52 pounds, precipitating 1.42 pounds $CaCO_3$.*

---

* I have received from the Marsac mill some data concerning silver precipitated from wash-water and weak solution, and from the regular lixiviation-solution; also, the values of these precipitates in silver and their percentages of lead and copper. According to these figures, the consumption, per ton of ore, of caustic soda in the manufacture of Russell's sulphide should be much less than it actually is, unless it be assumed that more than one-half of the Na_2S in Russell's sulphide is oxidized by contact with air, a supposition with which the figures agree remarkably well.

A sodium sulphide solution containing Na_2S, if allowed to oxidize, is unprofitable for another reason. Equation No. 2 shows that only one-half of the Na_2S is converted into $Na_2S_2O_3$, the other half becoming NaHO. Thus oxidation of the solution is unsatisfactory in two ways.

It has been shown, in § 603, that sodium hyposulphite is obtained in most cases at less expense by allowing the sodium sulphide solution to oxidize than by buying the hypo-salt; hence, it seems timely to abolish Russell's sulphide altogether and substitute in its place Na_2S_2, or even a higher polysulphide, if Na_2S_2 cannot be obtained free from Na_2S.

All the statements made above are supported by actual experiments. I prepared from chemically pure reagents two sodium sulphide solutions, one with sufficient sulphur to contain $Na_2S+Na_2S_2$, another to contain Na_2S_2. As I am fully aware, it is very difficult to prepare such solutions on a laboratory-scale, free from caustic soda or from higher polysulphides; but this does not influence in the least the principal points at issue.

The following tests were made with the freshly prepared sulphide solutions:

Solutions of calcium chloride and of gypsum in sodium hyposulphite were prepared, of such concentration that caustic soda produced in them a precipitate of calcium hydrate.

To these solutions the sodium sulphides were now added. Each drop of the concentrated reagents produced a slight flocculent precipitate of intensely yellow color, which would disappear again upon stirring. After successive additions of the reagents, the precipitates finally remained, but to effect this, much more of the solution containing Na_2S_2 was consumed than of that containing $Na_2S+Na_2S_2$. It became at once apparent that only a very small portion of the reagent was engaged in the reaction, the solution assuming an intensely yellow color. Upon adding a sufficient quantity of water, the precipitates dissolved again; hence, it was not possible to examine them further by filtering and washing with water. This was done, however, with alcohol, in which sodium polysulphides are soluble and calcium hydrate is not. After repeated washing with alcohol, the precipitates became white and proved to be calcium hydrate.

Now, 50 c. c. of each sulphide solution were put into 7-pound acid-bottles and shaken for three hours, air and carbonic acid being blown from the lungs into the bottles every few minutes. In the beginning, the solutions remained perfectly clear; finally, a small amount of sulphur separated, indicating the presence of higher polysulphides than Na_2S_2.

The oxidized sulphides were now tested with calcium solutions as before. Yellow precipitates appeared at once; but with equal quantities of the reagents, the precipitate from the Na_2S_2 solution was quite small compared with that obtained by the $Na_2S+Na_2S_2$ solution. These precipitates were not flocculent, settled quickly and did not dissolve upon addition of water. After washing, they became perfectly white, and proved to be $CaCO_3$. Q. E. D!

It is claimed that the precipitate of calcium carbonate can be avoided if precipitation of silver, copper, and lead is not done too closely. I am of an entirely different opinion. If the precipitate were actually an insoluble calcium sulphide, yes; but since it is $CaCO_3$, its precipitation must commence at once. The precipitate becomes apparent only after the above named metals have been completely precipitated, and if calcium still remains in solution. The precipitate appears yellow, because it holds mechanically sodium sulphide. Only prolonged washing makes it white. As will be seen from the example quoted previously, much more calcium may be in a lixiviation-solution per ton of ore than can be precipitated even by a well oxidized Russell's sulphide prepared from 6 pounds of caustic soda.

At the Aspen mill, according to W. S. Morse, sulphides were produced which contained, exceptionally, as much as 13 per cent. $CaO=23.2$ per cent. $CaCO_3$. The sodium sulphide used was the commercial Na_2S in crystals. Such a solution, as shown above, is liable to contain considerable Na_2CO_3; hence the large percentage of lime in the sulphides.

§ 605. *Regeneration of Sodium Hyposulphite from Tetrathionate.*

In preparing Russell's extra-solution, one pound of sodium hyposulphite is changed to tetrathionate for each pound of copper sulphate consumed. The formation of tetrathionate is further increased by atmospheric decomposition of extra-solution, so that, finally, the solution must hold a considerable amount of sodium tetrathionate, which salt is not a solvent for silver-compounds. Fortunately, a very simple reaction converts the tetrathionate again into hyposulphite:

$$Na_2S_4O_6+Na_2S_x=2Na_2S_2O_3+xS.$$

This reaction takes place in precipitating the solution.

So long as metals remain unprecipitated, it is doubtful whether this regeneration can take place. Close precipitation and over-precipitation, however, would give to the reaction free scope. Hence, it will be advantageous to precipitate closely, and to over-precipitate occasionally, the solution in a number of precipitating-tanks. The excess of sodium sulphide is then neutralized by running some fresh silver solution into the precipitating-tank. That all this is not merely theoretical speculation, is proved by statistics from the Yedras mill, Mexico. With a consumption of 7 pounds copper sulphate per ton of ore for making extra-solution, the loss in sodium hyposulphite, by close precipitation, was only one pound per ton of ore treated. When the consumption of copper sulphate was reduced to 3.85 pounds, the loss in hyposulphite fell to 0.74 pounds. In this case over-precipitation was not practiced; if it had been done, the loss in hyposulphite might have been reduced still further. That it is profitable to regenerate hyposulphite from tetrathionate, the following calculation will show, based upon quality and prices of chemicals assumed in § 603.

For the production of 100 pounds sodium hyposulphite, a quantity of Na_2S_2 would be needed, requiring:

21.5 pounds caustic soda	$1.18
16.3 pounds sulphur	0.57
Total cost	$1.75
100 pounds $Na_2S_2O_3+5$ aq.	$4.00
Net saving	$2.25

In this estimate the hyposulphite contained in and added by freshly prepared Na_2S_2 is included. In regenerating hyposulphite from tetrathionate, free sulphur is added to the precipitated sulphides, as follows:

For 100 parts hyposulphite regenerated, there are added:

When Na_2S is used	6.45	parts of sulphur
" Na_2S_2 "	12.90	" "
" CaS_5 "	32.25	" "

In these figures the hyphosulphite added by and contained in the sulphides is not included.

§ 606. *Precipitating-Coefficients.*

The precipitating-coefficients of a sulphide solution are the quantities of silver, copper, and lead precipitated for 100 parts of caustic soda or caustic lime consumed in its manufacture. In the same way precipitating-coefficients can be established for sulphur.

If a metal is precipitated by an alkaline polysulphide, RS_x, one equivalent of the latter precipitates one equivalent, or a double equivalent, of the former, $(x—1)S$ being liberated as free sulphur.

The following facts are of importance in reference to precipitating-coefficients for caustic soda. If the lower sodium polysulphides are prepared either from lye of proper concentration, but not of sufficient temperature before adding the sulphur, or from dilute lye by boiling it with sulphur, reagents may result with precipitating-coefficients for caustic soda far below the normal values recorded in tables a. and b. This can only be explained by assuming that a part of the caustic soda remains free and uncombined with sulphur, which is actually the case. For this and other reasons already stated, it is not advantageous to attempt the preparation of a solution containing Na_2S in perceptible quantity in order to save sulphur. A solution in which Na_2S largely predominates will always have precipitating-coefficients below the normal values for caustic soda.

On the other hand, very concentrated solutions of the higher polysulphides may give precipitating-coefficients far above normal values. Since, however, these abnormal values are only obtained by partial precipitation of silver, copper, and lead from a hyposulphite solution, and do not exist in complete precipitation, and consequently not in practical mill-work, the subject is without economical value, and will not be discussed here.

Russell's experiments on precipitating-coefficients—not knowing that they referred to partial precipitation only—misled me in the first edition of this book to build upon them an elaborate theory, which I herewith pronounce utterly worthless from a practical standpoint.

Although great accuracy cannot be claimed for the tables below, regarding actual mill-work, they will be sufficiently near the truth to base calculations of practical value upon them. They refer to freshly prepared sodium sulphide, and not to solutions oxidized by long contact with the atmosphere.

The following precipitating-schemes may be considered of practical value.

A. Lead and calcium are absent, or only present in very small quantities.

Method 1.—Precipitation by Na_2S_2, or CaS_5.

B. Lead and calcium are present in perceptible quantities, or lead alone is present.

Method 2.—Precipitation of lead by caustic lime, followed with precipitation of silver and copper by CaS_5.

Method 3.—Precipitation of lead and calcium by Solvay soda, followed with precipitation of silver and copper by Na_2S_2.

C. Calcium is present in considerable quantity, but not much lead.

Method 1, as given above, is here applicable; but in naming Na_2S_2 as precipitant under "C," it is distinctly understood that the reagent should be practically free from Na_2S, so that an oxidized solution contains only a minimum of Na_2CO_3. If it cannot be prepared with the amount of sulphur needed for the preparation of Na_2S_2 according to equation No. 1, more sulphur must be used.

In considering the economy of the different methods of precipitation, it would be misleading to rely only upon the figures given in the tables. There are other important elements that must enter into the calculation, outside of the cost of chemicals, and the relative proportions of silver, copper, lead, and calcium in the solution. It is important, for instance, whether the lead-calcium carbonate can be profitably sold. If calcium predominates, the product may be exceedingly low in lead, and only salable at the expense of the silver it contains. An equally important factor is the grade of the sulphides in silver. The lower the grade, the heavier the freight and the smelting-charges per ounce of silver, or the cost of refining if sulphides are treated at the mill. To this must be added increased expense in handling, namely, pressing, drying, sampling, and packing of precipitates. Evidently, general rules cannot be established regarding the most economical method of precipitation, but a calculation must be made for each individual case.

As repeatedly stated, tables for CaS_5 cannot be calculated on merely theoretical grounds. At attempt, however, is made in tables Nos. 606.e. and 606.f., to base values for CaS_5 on actual mill-statistics.

A comparative test was made between Russell's sulphide and CaS_5, at the Cusihuiriachic mill, Mexico, as follows:

The mill, reducing 50 tons of ore per day, was run 38 days, using Russell's sulphide, and 21 days, using CaS_5, as precipitant. The quantities of chemicals consumed were as follows:

Using Russell's Sulphide:

Average value of ore: 35.1 ounces silver per ton.
Consumption of caustic soda: 4.4 pounds per ton of ore.
Consumption of sulphur: 2.9 pounds per ton of ore.

Using CaS_5:

Average value of ore: 39.0 ounces of silver per ton.
Consumption of caustic lime: 24 pounds per ton of ore.
Consumption of sulphur: 10.3 pounds per ton of ore.

Outside of a difference in value, the ore was of exactly the same character in both cases. For better comparison, I reduce the above figures for CaS_5 to what they would have been in working ore of 35.1 ounces silver per ton, with the following result:

Corrected Values, Using CaS_5:

Consumption of caustic lime: 21.6 pounds per ton of ore.
Consumption of sulphur: 9.3 pounds per ton of ore.

To make these statistics more available for comparison, the composition of the sulphides should have been ascertained, besides the consumption in hyposulphite. Under the circumstances, I have to take the figures as they are. Considering what has been said about the preparation of CaS_5, and its deterioration by oxidation, the much higher consumption of chemicals involved in manufacturing CaS_5, compared with Russell's sulphide of equal efficiency, is by no means extravagant. To make comparison perfectly fair, we should add, however, for CaS_5 a gain of about 20 per cent. in hyposulphite, on a basis of the quantity of this salt contained in freshly prepared Russell's sulphide. For convenience in calculation, the addition of calcium hyposulphite is put down as its equivalent of the sodium-salt.

In the financial table No. 606.f., (see below), the price of caustic lime is taken at $\frac{3}{4}$ cent per pound, and other chemicals as stated in table No. 606.d.

It is interesting to compare table No. 606.f. with table No. 606.d. Even under favorable assumptions for CaS_5, this reagent is, at stated prices for caustic soda, sulphur, and sodium hyposulphite, very much dearer than Na_2S_2.

CaS_5 would be only cheaper where caustic lime can be obtained at much less than $\frac{3}{4}$ cent per pound, and where the cost of sulphur is abnormally low compared with that of caustic soda.

For this reason, in examining different methods of precipitation for a special case under normal conditions, we may just as well pay no attention to CaS_5 in Method 1, and discard Method 2 altogether.

Finally, if in lixiviation a considerable amount of sodium hyposulphite is consumed to keep up the strength of the stock-solution, it will be more profitable, in most cases, to use an oxidized sodium sulphide solution in place of one freshly prepared. Although the cost of precipitation is thereby apparently increased, the total cost of lixiviation will be diminished.

It has been claimed that in using CaS_5, the sulphides precipitate quicker and settle better than with sodium sulphide. This claim is not sustained by practical experience.

We find it stated in treatises on metallurgy that calcium hyposulphite is a more energetic solvent for gold than the sodium-salt. This statement is without foundation in theory or practice. The solvent energy of calcium hyposulphite for silver-compounds is slightly inferior to that of the sodium-salt for equal weights.

The difference in deterioration of the two hyposulphites by atmospheric influences will be given in § 608.

Table No. 606. a.—*Precipitating-Coefficients for c. p. Reagents.*

Calculations are based upon the following approximate chemical equivalents, frequently used :

$$H = 1;\ O = 16;\ C = 12;\ S = 32;\ Na = 23;$$
$$Ca = 40;\ Ag = 108;\ Pb = 207;\ Cu = 63.$$

Precipitating-Coefficients for NaHO, Consumed in the Manufacture of Na_2S_x.

100 NaHO precipitate 180.0 Ag as Ag_2S.
100 NaHO precipitate 172.5 Pb as PbS.
100 NaHO precipitate 105.0 Cu as Cu_2S.
Adding 103.3 $Na_2S_2O_3 + 5$ Aq.

The following quantities of NaHO and S, consumed in the manufacture of Na_2S and Na_2S_2, are needed for the precipitation of metals:

For 100.	NaHO.	Sulphur for		Adding $Na_2S_2O_3 + 5$ aq.
		Na_2S.	Na_2S_2.	
Ag.	55.5	29.6	44.4	57.3
Pb.	58.0	30.9	46.4	59.9
Cu.	95.2	50.7	76.1	98.3

Precipitation of Lead and Calcium by Na_2CO_3.

100 Na_2CO_3 precipitate 195.3 Pb as $PbCO_3$.
100 Na_2CO_3 precipitate 37.7 Ca as $CaCO_3$.
For precipitation of
100 Pb are needed 51.2 Na_2CO_3.
100 Ca are needed 265.0 Na_2CO_3.

TABLE NO. 606. b.—*Precipitating-Coefficients for Commercial Reagents.*

Caustic soda with 90 per cent. NaHO, corresponding to the English rating of 70.6 per cent.; sulphur with 95 per cent. S; Solvay soda with 98 per cent. Na_2CO_3.

Precipitating-Coefficients for Caustic Soda, Consumed in the Manufacture of Na_2S_x.

100 Caustic soda precipitate 162.0 Ag as Ag_2S.
100 Caustic soda precipitate 155.2 Pb as PbS.
100 Caustic soda precipitate 94.5 Cu as Cu_2S.
Adding 93 $Na_2S_2O_3+5$ aq.

The following quantities of caustic soda and sulphur, consumed in the manufacture of Russell's sulphide and Na_2S_2, are needed in the precipitation of metals:

For 100.	NaHO.	Sulphur for		Adding $Na_2S_2O_3+5$ aq.
		Russell's Sulphide.	Na_2S_2.	
Ag.	61.6	41.0	46.7	57.3
Pb.	64.4	42.9	48.8	59.9
Cu.	105.7	70.4	80.0	98.3

Precipitation of Lead and Calcium by Solvay Soda.

100 Solvay soda precipitate 191.4 Pb as $PbCO_3$.
100 Solvay soda precipitate 36.9 Ca as $CaCO_3$.
For precipitation of
100 Pb are needed 52.2 Solvay soda.
100 Ca are needed 270.3 Solvay soda.

TABLE No. 606. c.—*Weights of Precipitates and their Percentages in Metals.*

Weights of Sulphides.

From 100.	If precipitated by		
	Russell's Sulphide.	Na_2S_2.	CaS_5.
Ag.	124.3	129.6	174.1
Pb.	125.3	130.9	177.3
Cu.	141.6	150.8	227.0

Weights of Carbonates.

The precipitation by Solvay soda will produce from:

100 Pb, ...	129.0 $PbCO_3$
100 Ca, ...	250.0 $CaCO_3$

Percentages of Metals in Sulphides.

Percentage of	If precipitated by		
	Russell's Sulphide.	Na_2S_2.	CaS_5.
Ag in Silver Sulphide	80.4	77.1	57.4
Pb " Lead "	79.7	76.4	56.4
Cu " Copper "	70.6	66.3	44.0

Percentage of Metal in Carbonates.

Lead carbonate, 77.5 Pb.

Table No. 606. d.—*Financial Results.*

Showing the cost of precipitation by Russell's sulphide, Na_2S_2, and by Solvay soda, assuming the following prices for commercial chemicals put down at the mill: caustic soda, 5.5 cents; sulphur, 3.5 cents; Solvay soda, 4.0 cents; hyposulphite, 4.0 cents per pound.

Cost of Precipitating Metals as Sulphides.

For 100 Pounds.	By Russell's Sulphide.	By Na_2S_2.	Gain in $Na_2S_2O_3$ + 5 aq.	Net Cost using.		Difference in cost between Russell's Sulphide and Na_2S_2.
				Russell's Sulphide.	Na_2S_2.	
Ag.	$4.82	$5.02	$2.29	$2.53	$2.73	$0.20
Pb.	5.04	5.25	2.40	2.64	2.85	0.21
Cu.	8.28	8.61	3.93	4.35	4.68	0.33

Cost of Precipitating Lead and Calcium as Carbonates.

For 100 pounds Pb., $2.09
For 100 pounds Ca., 10.81

Table No. 606. e.—Showing the quantities of caustic lime and sulphur consumed in the manufacture of CaS_5, needed for the precipitation of metals.

For 100 Pounds.	Caustic Lime.	Sulphur.	Adding $Na_2S_2O_3+5$ aq.
Ag.	302.4	130.2	68.8
Pb.	316.1	136.1	71.9
Cu.	519.0	223.4	118.0

This table is based on comparative statistics obtained in the Cusihuiriachic mill, Mexico.

TABLE No. 606. f.—Showing the cost of precipitation by CaS_5, assuming price of caustic lime at $\frac{3}{4}$ cents per pound, and that of other chemicals the same as given in table No. 606. d.

Cost of Precipitating 100 Pounds.	Caustic Lime.	Sulphur.	Total.	Gain in $Na_2S_2O_3+5$ aq.	Net Cost.
Ag.	$2.26	$4.56	$6.82	$2.75	$4.07
Pb.	2.37	4.76	7.13	2.88	4.25
Cu.	3.89	7.82	11.71	4.72	6.99

This table is based upon the figures given in table No. 606. e.

§ 607. *Application of the Tables to Special Cases.*

In this paragraph the tables will be applied to examine the cost of precipitation in special cases. It is assumed that the ore contains lead and a small amount of calcium. Although its contents in copper are assumed insignificant, the copper from extra-solution, necessary for treatment, comes to precipitation. We will neglect the comparatively small amount of silver precipitated with the lead-calcium carbonates, assuming that such silver is paid for at the same rates as silver in sulphides. Prices of chemicals are taken as stated in table No. 606. d.

We further assume the following terms, according to a case as it really exists, for the disposal of the sulphides to smelting-works and the cost of handling the precipitates:

The Marsac mill sells its sulphides to the Omaha & Grant S. & R. Co., the latter paying for 97 per cent. of the silver, New York quotations, charging $30 per ton for treatment, but paying freight from Park City to Omaha. No allowance is made for copper or lead. [Gold is paid for at the rate of $20 per ounce.]

Carbonate precipitates are taken at the same rates, as also the low-grade precipitates from the first wash-water.

The cost of handling precipitates (sampling, sacking, and hauling to railroad depot) is $10 per ton.

EXAMPLE.

The solution obtained in lixiviation contains per ton of ore:

3 pounds (43$\frac{3}{4}$ ounces) silver.
1 pound copper.
3 pounds lead.
$\frac{1}{2}$ pound calcium.

Four pounds sodium hyposulphite have to be regenerated from tetrathionate in precipitation. This expense, however, we need not consider, since it is the same for Methods 1 and 3.

I hardly need remind the reader that in precipitation Na_2S_2 free from Na_2CO_3 is used, and not an oxidized solution of Russell's sulphide, which would also precipitate calcium in Method 1, changing the weight and grade of the sulphides.

Weights of Precipitates per Ton of Ore.	Precipitation by	
	Method 1.	Method 3.
Sulphides.	9.32 pounds.	5.40 pounds.
Carbonates.	-----------	5.12 "
Sulphides contain:		
Ag, per ton, oz.	9388.5	16,201.7
Cu, per cent.	10.73	18.52
Pb, " "	32.19	------------
Carbonates contain Pb, per cent. 58.6.		
Cost per Ton of Ore of	Cents.	Cents.
Precipitation.	21.42	25.45
Handling.	4.66	5.26
Reduction.	13.98	15.78
Total.	40.06	46.49

It will be seen at once that separate precipitation of lead and calcium cannot be profitable if carbonates are sold at sulphide-rates. Also, that Method 3 becomes more costly, the more calcium is present in proportion to lead.

We will now assume 90 per cent. of the lead in carbonates to be paid for at the rate of 2 cents per pound, and freight and smelting-charges to be $15 per ton; also that silver is paid for at sulphide-rates. This would give the following result per ton of ore:

	Cents.
Precipitation	25.45
Handling	5.26
Reduction, sulphides	8.10
Reduction, carbonates	3.84
Total	42.65
Less amount realized from sale of lead	5.40
Total	37.25

Should the buyer, however, deduct 5 per cent. of the silver in place of 3 per cent., nearly all the profit from the sale of lead may be consumed, or even a negative balance left, depending on the richness of the carbonates in silver.

It is hardly necessary to continue this discussion. The separate precipitation of lead and calcium by Solvay soda is evidently not so profitable as its advocates claim.* If lead alone is present, or the amount of calcium is very slight in proportion to lead, Method 3 can be used to advantage.

If sulphides are refined at the mill by a humid process with sulphuric acid, the absence of lead and calcium is essential to success; and in this case Method 3 is of great value.

§ 608. *The Loss of Hyposulphite in Lixiviation.*

In connection with precipitation, some remarks about the loss of hyposulphite in lixiviation will be pertinent.

As has been shown, large amounts of hyposulphite are added to the stock-solution in precipitation, even with freshly prepared calcium or sodium sulphides. From this it might be expected that the stock-solution would always increase in strength and concentration. This is only the case if, in ordinary lixiviation, an oxidized solution of CaS_5 is used as precipitant, and comparatively large amounts of copper and lead are thrown down together with the silver.

---

* According to calculations of W. S. Morse, the separate precipitation of lead was, for example, not profitable with Aspen ore. Taking into consideration the disposal and treatment of the precipitates, separate precipitation of lead increased the expense of precipitation from 19 to 23 per cent. as compared with using sodium sulphide alone.

The losses in hyposulphite are partly mechanical, and partly caused by chemical decomposition. The mechanical losses take place when the first wash-water is replaced by stock-solution, and again when the latter is replaced by the second wash-water. In these operations, no matter how carefully they are conducted, water and stock-solution are more or less mixed, especially where the filters are not in good condition and are partly choked. Thus, in replacing the first wash-water, a certain quantity of hyposulphite solution results, too weak to be mixed with the normal solution; for this reason it is transferred to a separate precipitating-tank, precipitated there by itself, and the decanted clear solution run to waste. As soon as the solution shows a certain strength in hyposulphite, which is ascertained by the iodine-test, it is turned into a regular precipitating-tank. The same is the case in replacing the solution by the second wash-water. The higher the percentage of hyposulphite in the stock-solution, the greater this mechanical loss.

The chemical losses are caused as follows:

1. By oxidation in contact with the atmosphere, hyposulphite being converted into sulphate. This loss depends on the quantity of solution used per ton of ore and kept in rotation, its temperature, and upon atmospheric conditions, most likely the contents of the atmosphere in ozone.

Russell found that a stock-solution, not used for some time, had deteriorated as follows:

Solution not used for twenty-six days.

100 c.c. dissolved,	0.485 gm. silver as AgCl.
After twenty-six days,	0.458 gm. “ “
Deterioration,	5.5 per cent.

Solution not used for thirty-five days.

100 c.c dissolved,	0.447 gm. silver as AgCl.
After thirty-five days,	0.359 gm. “ “
Deterioration,	17.4 per cent.

In order to test the relative deterioration of solutions of calcium and sodium hyposulphite, Russell carried out the following experiments:

Solutions of different concentration were exposed in soup-plates to the atmosphere for seven days at a temperature of from 20° to 22° C., and brought to their original volume at the expiration of that time. The relation of the depth of the solution to the diameter of the plate was about 1 to 8. A re-determination of the solvent energy of these solutions for silver chloride gave the following average results, namely: The sodium hyposulphite solutions had deteriorated $1\frac{4}{10}$ per cent.; the calcium solutions had deteriorated $16\frac{1}{10}$ per cent.*

2. By converting hyposulphite into tetrathionate in preparing extra-solution, and further formation of tetrathionate by atmospheric oxidation of extra-solution. This loss is, however, made largely good again by close precipitation, and may be almost completely covered by over-precipitation.

3. By decomposition of extra-solution with formation of Cu_2S and sulphates; this loss being final and irretrievable. It is greatest where extra-solution has to perform much work—*i. e.*, where standard extra-solution must be used warm and is circulated.

---

* It would have been more correct if Russell had determined the deterioration of the hyposulphite solutions by the iodine-test.

CHAPTER VII.

LABORATORY WORK.

In a lixiviation-mill the following determinations should be made every day:

1. Pulp-assay of the ore and salt mixture.
2. Determination of the percentage of salt in the pulp.

From these data the value of the ore is calculated.

3. Pulp-assay of the roasted ore.
4. Determination of the percentage of soluble salts in the roasted ore.

From these data the value of the roasted ore, minus soluble salts, is calculated. (The so-called "calculated value" of the roasted ore.)

5. Tailings-assay.

In comparing the value of the tailings with the value of the roasted ore, minus soluble salts, the so-called "apparent" extraction is found.

6. Lixiviation-tests of the roasted ore with ordinary solution.
7. Lixiviation-tests of the roasted ore with extra-solution.

By comparing No. 6 with No. 7, we learn how much the extraction of the silver will be benefited by the use of Russell's process.

If the ore is treated raw, most of these determinations are not required.

Silver in wash-water, in treating roasted ores, is determined by precipitating a measured quantity with sodium sulphide, collecting the precipitate on a filter, and assaying it for silver.

Of other work, there is the volumetric determination of sodium hyposulphite in the stock-solution, and testing ores in the laboratory regarding their fitness for lixiviation.

§ 701. *Determination of the Percentage of Salt in Crushed Ore.*

To determine the percentage of salt, take 10 gm. of the pulp, put it on a filter, and leach with hot water until the filtrate ceases to give a reaction with silver nitrate. For drying of the residue, the funnels are placed in a drying-chamber, heated by steam or boiling water. The dry residue is removed from the filter with a brush and weighed.

§ 702. *Determination of Soluble Salts in Roasted Ore.*

The same "modus operandi" can be adopted as in § 701; subsequent treatment with sodium hyposulphite solution not being absolutely necessary. In most cases the result of this determination will agree sufficiently close, say within 1 per cent., with the percentage of soluble salts actually extracted in the mill. If, however, the roasted ore contains considerable calcium sulphate and caustic lime, both of which are not easily soluble either in water or in sodium hyposulphite solution, the laboratory-test will indicate a much higher solubility than that reached in the mill, and a correction must be made. The reason for this discrepancy is, that in operating on a large scale, the calcium sulphate and the caustic lime are only partially removed, owing to the much smaller quantity of solution used in proportion to the ore than in the laboratory-test. The correction is made as follows, viz: 10 gm. of the roasted ore, and the same amount of the tailings corresponding to the ore, are placed on filters. Each is leached first with about 300 c. c. of cold water, and then with a cold sodium hyposulphite solution of about 10 per cent. concentration, until the filtrates cease to give a reaction with sodium carbonate, indicating that all the calcium-compounds are extracted. Water and sodium hyposulphite solution, if cold, it will be remembered, are better solvents for calcium hydrate and sulphate than if hot. The residues are washed with water, dried and weighed. The difference between the percentage of solubility of the ore, and that of the tailings, gives, with sufficient exactness, the percentage of salts extracted in the mill by both water and lixiviation-solution. The slight error is due to the fact that the ore and tailings are not of the same nature—the latter having been concentrated, previously to the weighing, by an amount equal to the solubility in the mill; but the determination is sufficiently exact for practical purposes.

§ 703. *Lixiviation-Test with Ordinary Solution.*

The quantity of ore used is ½ A. T. Roasted acid ores are first leached with 300 c. c. water. After decanting the clear wash-water, 250 c. c. of a 5 per cent. sodium hyposulphite solution are added to the ore in the beaker, and the latter is heated on a sand-bath until a temperature of 55° C. has been reached. Stirring the solution frequently, it is kept at that temperature for half an hour. Now, the solution is decanted on the same filter to which the tailings are subsequently transferred.* The filter, after washing the tailings with water, is manipulated as in § 701, and the tailings assayed. The tedious operation of burning the filter is entirely unnecessary, the small amount of dust, that may be left on the paper, not being sufficient to influence the result in a perceptible degree.

The method of making the ordinary lixiviation-test, as described above, will suffice in most cases. Some ores, however, require a different treatment. This is especially the case with roasted ores containing considerable lime. Such ores should not be leached with water, but treated at once with hyposulphite solution. At Yedras, the ordinary lixiviation-test is not made in beakers, but the sample is treated on a filter with a boiling sodium hyposulphite solution, until the filtrate is free from silver.

§ 704. *Influence of re-pulverizing Roasted Ore in an Iron Mortar.*

In case roasted ores have been so coarsely crushed in the mill as to require a re-pulverizing for assaying, an iron mortar should not be used. As shown in Table No. 704, the extraction of silver from Cusi ore, by ordinary solution in the assay-office, is reduced 10 to 11 per cent., while results with extra-solution are practically unaltered. This is caused by reduction of silver to the metallic state, in which it is much more soluble in extra-solution than in ordinary solution.

On Lake Valley ores, the use of the iron mortar also reduced the extraction by extra-solution 18 per cent.

---

* In some mills a hyposulphite solution of higher concentration than 5 per cent. is used, and the time of treatment is longer.

TABLE No. 704.

Experiments on the Effect of Grinding Roasted Ore in Wedgewood or Agate, and in Iron Mortars.

Ore from:	Mesh of Screen and Method of Crushing.	Sample from:	Material of Mortar.	Value of Ore.	Value of Tailings from Ordinary Solution.	Value of Tailings from Extra-Solution.	Per cent. Extracted by Ordinary Solution.	Per cent. by Extra-Solution.
				Ounces per ton.				
Cusi.	No. 20-Screen. Stamps.	Furnace	Wedgewood.	30.2	6.4	2.8	78.9	90.7
			Iron.	30.2	9.6	2.8	68.3	90.9
		Ore-tank.........	Agate.	30.0	4.4	2.6	85.3	91.3
			Iron.	30.0	7.4	2.8	75.4	90.3
Lake Valley.	No. 16-Screen. Rolls.	Furnace	Agate.	16.0	2.9	2.0	81.3	87.5
			Iron.	16.0	6.7	6.0	58.1	62.5
		Cooling-floor ...	Agate.	18.0	3.7	3.5	79.4	80.6
			Iron.	18.0	7.2	6.0	66.0	66.7
		Ore-tank.........	Agate.	18.0	3.0	3.0	83.3	83.3
			Iron.	18.0	9.0	5.7	50.0	68.3

§ 705. *Influence of Copper Vessel.*

It is customary, in some mills, to heat the sodium hyposulphite solution, used in the assay-office, in a copper vessel. Copper is dissolved, as shown in § 206, and a weak extra-solution formed. According to F. Johnson's observations, the effect of this extra-solution may be considerable, as shown in Table No. 705.

TABLE No. 705.

Effect on the Lixiviation-Tests with Ordinary Solution, by Heating the Latter in a Copper Vessel.

Value in Ounces per Ton.			Ounces in Favor of Copper Vessel.	Per cent. in Favor of Copper Vessel.
Roasted Ore.	Tailings from Solution heated in Copper Vessel.	Tailings from Solution heated in Glass Vessel.		
75.60	9.84	11.52	1.68	2.2
76.32	9.84	12.90	3.06	4.1
63.96	14.94	16.82	1.88	3.0
69.30	15.96	18.60	2.64	3.8
72.78	8.40	9.12	1.02	5.4
72.72	8.28	10.98	2.11	3.7

Lixiviation-Tests with Extra-Solution.

In the following paragraphs the methods of making lixiviation-tests with extra-solution are given. One or more of these methods will be found to be suitable for any class of ore, and will show, very approximately, what extraction can be obtained on a large scale in the mill. These methods are entirely empirical and the result of numerous tests with ores of great variety in character.

For the sake of brevity, the terms "Hypo." and "Blue" are used for sodium hyposulphite crystals, and a blue-stone solution containing 20 per cent. of $CuSO_4 + 5$ aq.

§ 706. *Lixiviation-Tests with Extra-Solution for Raw Ores and Tailings.*

No. 1. Add 250 c. c. cold water and 20 gm. Hypo; let stand 12 to 16 hours cold; decant; add 25 c. c. Blue; dilute to 300 c. c. with cold water; let stand 12 to 16 hours cold; add 20 gm. Hypo; let stand 12 to 16 hours cold; filter and wash.

No. 2. Add 250 c. c. cold water and 20 gm. Hypo; let stand 12 to 16 hours cold; add 20 gm. Hypo and 25 c. c. Blue; let stand 12 to 16 hours cold; filter and wash.

No. 3. Add 100 c. c. cold water and 25 c. c. Blue; let stand 12 to 16 hours cold; dilute to 300 c. c. with cold water; add 20 gm. Hypo; let stand 12 to 16 hours cold; filter and wash on filter with hot Hypo solution.

No. 4. Add 100 c. c. cold water and 25 c. c. Blue; let stand 1 hour cold; add 200 c. c. cold water and 20 gm. Hypo; heat to about 55° C.; filter and wash.

No. 5. Add 250 c. c. cold water and 25 c. c. Blue; let stand 1 hour cold; add 20 gm. Hypo; heat on sand-bath to about 55° C.; decant; add 200 c. c. cold water and 25 c. c. Blue; let stand 1 hour cold; add 20 gm. Hypo; heat, filter and wash.

No. 6. Add 250 c. c. cold water and 25 c. c. Blue; heat to about 55° C.; decant; add 20 gm. Hypo and 25 c. c. Blue; dilute to 300 c. c. with cold water; heat to about 55° C.; filter and wash.

No. 7. Add 300 c. c. cold water and 30 gm. Hypo; heat to 44° C.; filter, wash with water, break filter and wash contents through into beaker; add 25 gm. Hypo and 25 c. c. Blue; dilute to 300 c. c. with cold water; let stand 20 hours; decant on filter; add 25 gm. Hypo, 25 c. c. Blue and 300 c. c. cold water; heat to 44° C.; filter and wash.

§ 707. *Lixiviation-Tests with Extra-Solution for Roasted Ores and Tailings.*

No. 8. No washing; add 50 c. c. cold water and 25 c. c. Blue; let stand 2 hours cold; dilute to 300 c. c. with cold water; add 20 gm. Hypo; let stand 2 hours cold; filter and wash.

No. 9. No washing; add 250 c. c. cold water and 25 c. c. Blue; let stand 12 to 18 hours cold; add 20 gm. Hypo; let stand 12 to 18 hours cold; filter and wash.

No. 10. No washing; add 250 c. c. cold water and 25 c. c. Blue; let stand 2 hours cold; add 20 gm. Hypo; let stand 2 hours cold; filter and wash.

No. 11. Wash with cold water; add 25 c. c. Blue; dilute to 300 c. c. with cold water; let stand 12 to 18 hours cold; add 20 gm. Hypo; let stand 12 to 18 hours cold; filter and wash on filter with hot Hypo solution.

No. 12. No washing; add 250 c. c. cold water and 25 c. c. Blue; let stand 1 hour cold; add 20 gm. Hypo; heat to about 55° C.; decant; add 250 c. c. cold water and 25 c. c. Blue; let stand 1 hour cold; add 20 gm. Hypo; heat to about 55° C.; filter and wash.

No. 13. Wash with hot water; add 20 gm. Hypo; let stand with residue of wash water, about 30 to 40 c. c., 1 hour; add 25 c. c. Blue; dilute to 300 c. c. with cold water; heat to about 55° C.; filter and wash.

No. 14. Wash by decantation with cold water; add 300 c. c. cold water and 30 gm. Hypo; heat to 44° C.; filter and wash with water; break filter and wash contents through into beaker; add 25 gm. Hypo and 25 c. c. Blue; dilute to 300 c. c. with cold water; let stand 20 hours; decant into filter; add 25 gm. Hypo, 25 c. c. Blue and 300 c. c. cold water; heat to 44° C.; filter and wash.

§ 708. *Control of the Mill-Work by Laboratory-Tests essential in Lixiviation.*

To a novice in lixiviation, this laboratory-work looks much more formidable than it is in reality. It has to be considered that in working ore of a known character the severe work of many tests is only required in starting the mill. After the laboratory-test—corresponding most nearly to the best treatment on a large scale—has once been selected, the matter becomes simple enough. A close control of the mill-work, by tests in the laboratory, is much more essential in lixiviation than in treating ores by amalgamation, since the later process, especially with roasted ores, can only be subjected to a few changes as compared with the former. In making the so-called chlorination-tests of roasted ore in a pan-mill, we obtain a control of the work performed by the roasting-furnaces, and nothing else. In many cases there is a close correspondence between the chlorination-test and the extraction by amalgamation; in other cases no such agreement exists. An ore may show a high chlorination, and give comparatively low results in amalgamation. This is often due to the presence of silver antimonate and arsenate, which are, as we have seen, extracted by ordinary hyposulphite solution, but are less amenable to amalgamation. From such tailings a large percentage of the silver can be extracted by lixiviation. It also occurs frequently that amalgamation shows a higher yield than the chlorination-test, and even then, silver remains in the tailings that can be extracted by lixiviation. In this instance, the difference may be due to metallic silver, and to other causes not known.

§ 709. *Volumetric Determination of Sodium Hyposulphite with Iodine.*

The method is based upon the reaction:

$$2\,Na_2S_2O_3 + 2\,I = 2\,NaI + Na_2S_4O_6.$$

and the property of free iodine to turn starch blue. Hence, if to a hyposulphite solution, containing starch, iodine is added, the blue color of the starch will appear as soon as all the hyposulphite has been changed to tetrathionate.

If c. p. iodine cannot be obtained, the commercial article should be purified. This is done by mixing the iodine with [about $\frac{1}{6}$ of its weight] potassium iodide, and slowly heating the mixture in a large watch-glass, with ground rim, on a sand-bath. As soon as violet vapors appear, the watch-glass is covered by a second one of equal diameter. Heating is continued until the iodine has sublimated. Chlorine and bromine, frequently found in common iodine, remain combined with potassium in the residue of potassium iodide.

The potassium iodide, used for dissolving the iodine, should be pure, and not contain free iodine. Its solution, made acid with c. p. hydrochloric acid, should not turn blue upon addition of starch-solution.

Preparation of the Iodine-Solution.—5.121 gm. iodine are placed in a litre-bottle with glass-stopper, together with 7.5 gm. potassium iodide, and 100 c. c. distilled water. A slight excess above this weight may be taken, as it is easier to correct the titre of a solution that is too strong than of one that is too weak. The iodine and potassium iodide are brought together in one spot, and the bottle is left in a dark and moderately warm place. If, after twenty-four hours, the iodine has not all dissolved, the solution is shaken, more potassium iodide is added, and the iodine is again collected in one spot so that it may come in contact with the potassium iodide crystals. After complete solution of the iodine has taken place, the litre-bottle is filled to the mark with distilled water and shaken. The solution is kept in bottles with glass-stoppers, and in the dark. Under these circumstances it will be quite permanent. 1 c. c. of this solution should correspond to 0.010 gm. of sodium hyposulphite.

Preparation of the Starch-Solution.—To one part of starch, mixed with a little water, 100 parts of boiling water are added, and the solution is allowed to settle. After decanting the clear solution the rest is filtered. Pure common salt is now added until saturation has taken place, and after the solution has become clear it is preserved in bottles of 3 or 4 ounces capacity. This solution will keep longer than one year. In place of salt, F. M. Watson adds a slight quantity of salicylic acid to the starch-solution. The latter method is better and more convenient than the former.

Determination of the Titre of Iodine-Solution.—0.100 gm. c. p. sodium hyposulphite is dissolved in 20 c. c. water, to which 1 c. c. starch-solution is added. Then the iodine-solution is added from a graduated burette, until the starch turns blue. The burette should have a glass-cock. If the titre of the iodine-solution is correct, 10 c. c. should have been consumed. If there is a perceptible difference, the titre of the solution must either be corrected, or the difference should be taken into account in testing the stock-solution.

Determination of the Concentration of the Stock-Solution in Hyposulphite.—Take, for instance, 10 c. c. of stock-solution, dilute to about 50 c. c., and proceed as above. If the titre of the iodine solution is correct, each 10 c. c. consumed represent 1 per cent. of sodium hyposulphite.

The stock-solution, before testing, should not be caustic, but either neutral or slightly acid. In case it is caustic, it should be neutralized with dilute sulphuric or hydrochloric acid. An excess of acid, however, must be avoided so as not to decompose hyposulphite salts.

§ 710. *Influence of Silver, Copper, and Lead on the Correctness of Results.*

It is important to know the effect of silver, copper, and lead present in a hyposulphite solution, if the latter is tested with iodine.* In the following experiments, 0.200 gm. $Na_2S_2O_3+5$ aq. were in each case dissolved in water, and then salts of silver, lead, and copper were added in such quantities that their contents in these metals were 0.005 gm. To the copper-solution, an extra amount of sodium hyposulphite was added so as to compensate for decomposition of this salt by copper sulphate, leaving exactly 0.200 gm. unchanged.

The silver-solution became slightly turbid at the end of the reaction, but the latter was quick and definite, recording exactly 0.200 gm. $Na_2S_2O_3+5$aq.

The lead-solution remained clear, the result being as accurate as above. If a large quantity of lead is present, the latter falls out as yellow iodide; hence, if hyposulphite is to be determined in such a solution, the lead should first be separated by sodium carbonate.

The copper-solution became very turbid, and the end-reaction was slow and indistinct. The sodium hyposulphite found was 0.225 gm. This is due to the presence of the copper in the form of a cuprous salt. If hyposulphite is to be determined in a solution containing copper, the following method may be used.

---

* In regular mill-work, *i. e.*, when precipitation of the metals is close, the small amount of copper present in the stock-solution does not affect results materially.

Take equal parts of the solution, and find in one hyposulphite plus cuprous oxide; in the other the amount of copper. The latter can easily be separated as Cu_2S by boiling the acid solution. For practical purposes it is sufficiently accurate to deduct for each 0.005 gm. copper, 2 c. c. iodine-solution, if the latter is of correct standard.

§ 711. *Assaying of Ores, Tailings, and Lixiviation-Products.*

In order to obtain correct and reliable metallurgical statistics in lixiviation, all values of ores, tailings, and products should be determined by "corrected" and not by "commercial" assays.* Commercial assays are those in which the direct result of a crucible- or scorification-assay is given; corrected assays are those in which the slags and cupels are assayed and the precious metals therein found are added to the result of the commercial assay. The table below shows the great difference between commercial and corrected assays for silver, and is based upon statistics obtained from seven lixiviation-mills. It shows that the percentage of incorrectness of the commercial assay increases inversely with the value of the ore or product.

Summary of the Differences Between Commercial and Corrected Assays of Ores, Tailings, and Lixiviation-Products.

Description of Material Assayed.	Grade of Value. Ounces per Ton.	Range of Inaccuracy. Per cent.
Tailings	1 to 20	21.8 to 8.4
Raw Ore	20 to 50	19.7 to 5.0
Washed Roasted Ore	15 to 35	16.8 to 9.0
Roasted Ore	20 to 45	10.6 to 5.4
Lead Carbonates	50 to 500	4.8 to 2.9
Base Sulphides	500 to 5000	5.1 to 2.2
Regular Sulphides	1000 to 16,000	4.9 to 1.2

The meaning of the last column, "Range of Inaccuracy," is that the commercial assay is so many per cent. lower than the corrected assay.

All assays of ores and tailings should be made in crucibles; those of lixiviation-products by scorification. Comparative tests, made in various mills, and with dry silver-ores of great variety in character, have demonstrated conclusively that the above rule gives the highest and most reliable results.

---

* For full information on this subject, see the author's paper: The Inaccuracy of the Commercial Assay for Silver, etc., in Vol. XXIV, Trans. Am. Inst.

In assaying ores and tailings it is not necessary that the corrected assay should be made every day for every sample. It is sufficient to make this correction for average weekly or even monthly samples.

§ 712. *Testing Ores regarding their Fitness for Lixiviation.*

The metallurgist is often called upon to give an opinion regarding the treatment of ores by various processes. In order to determine whether an ore can be treated with profit by lixiviation, he has to investigate the following points:

1st. Whether the ore can be lixiviated raw, or requires roasting.

2d. How fine the ore has to be pulverized.

3d. The percentage of salt required for chloridizing-roasting.

4th. The difference between lixiviation with ordinary and extra-solution.

5th. The quantity of base metals dissolved by hyposulphite solutions.

In case the ore can be treated raw, the investigation is simple enough, the fineness to which the crushing must be carried being the principal point at issue.

Ores requiring roasting cause much more work. The following hints will be useful to beginners. Roasting-tests are best carried out in the muffle of an assayer's cupelling-furnace, in clay-dishes of about 4½ to 5 inches diameter, which will conveniently hold a charge of 3½ A. T. In order to arrive at results with the least amount of labor, it is best to commence with charges of ore that have been crushed so as to pass a No. 40 screen, and mixed with a maximum percentage of salt, which may be taken between 10 and 15 per cent., according to the character of the ore. Desulphurizing should be done slowly, and at a low heat, under constant stirring. As soon as glowing particles cease to appear, and the charge commences to swell, the temperature of the ore is raised to a dull, red heat and so maintained for half an hour. Another charge is roasted for one hour, to see if the chlorination of the silver is thereby increased. Samples are now lixiviated with ordinary solution, and by methods with extra-solution that are most adapted to the character of the ore. An examination of the raw ore will *a priori* indicate if the roasted ore may be expected to contain caustic lime, while leaching with water will definitely show whether the wash-water is alkaline, neutral or acid. After the proper method of applying the extra-solution has been determined, and also the influence of time in roasting ascertained, new roasting- and lixiviation-tests are made with a reduced percentage in salt, until an economical limit is reached. These facts established, other roasting-tests are in order with ore crushed through coarser screens. If desirable, an investigation as to influence of temperature and time of roasting can be resumed. All the results are tabulated, so that they can be compared with each other at a glance.

It is not always possible to produce by muffle-roasting on a small scale the same effect that can be obtained by actual mill-work, especially if an ore is treated requiring banking-up on the cooling-floor for many hours in order to reach a high chlorination of the silver. Hence, an unfavorable result is not always a proof that the ore is unsuitable for lixiviation. Should a failure of these experiments be due to the absence of a sufficient quantity of sulphurets to decompose the salt, new roasting-tests should be made with the admixture of from 1 to 3 per cent., or more, of pyrites of iron, or half the quantity of sulphur. Of course, it should be ascertained beforehand if pyritic ores or sulphur can be procured, in a given locality, at a reasonable cost. There are cases in which an ore gives highest results if the roasting is accomplished in the shortest time possible, with a charge spread out so that its thickness does not exceed one-eighth of an inch.

PART II.

THE LIXIVIATION-PROCESS

IN ITS

PRACTICAL EXECUTION.

CHAPTER XI.

THE LIXIVIATION-PLANT.*

A modern lixiviation-plant consists of wooden tanks in which the ore is treated and solutions are accumulated; apparatus for elevating and transferring solutions, and for creating a vacuum below the filters of ore-tanks; filter-presses and drying-chambers for handling precipitates; apparatus for manufacturing sodium sulphide, etc.

§ 1101. *Wooden Tanks.*

Construction.—Tanks should be made of clear, well-seasoned lumber. In the United States, Oregon pine is the best material for this purpose. The staves, from 3 to 4 inches thick, according to size of tank, should be ordered cut to sweep of radius, and from 9 to 10 inches longer than the inside depth, but not "gained" for the bottom. The gaining of the staves, 1 inch deep, is done by hand, leaving a chine of 6 inches below the bottom. In all tanks the staves stand perpendicular to the bottoms. The bottom pieces, 3 to 4 inches thick, are cut to a diameter of 2 inches greater than that of the finished tank; they are grooved and joined by a tongue. All joints must be fitted with precision. White lead should never be put between the staves, but may be used in inserting the tongues between bottom pieces. The understructures of substantial timbers, placed on a solid foundation, should be sufficiently high to allow access to the bottom in case of leakage. The bottoms rest on joists, 3 to 4 inches wide and 10 to 12 inches deep, placed about 2 feet 6 inches apart, so that the staves are left entirely free. Hoops are made of round iron, ⅞ to 1⅛ inches diameter, the threaded ends, with hexagonal nuts, passing through forged or cast-iron lugs, giving preference to the former. In order to get the full strength of the rods, the threaded ends are taken ¼ inch larger than the diameter of the rod. For tanks of large diameter, each hoop is made in two or three sections; this is necessary to effect a more uniform closing of the stave-joints by tightening the nuts in two or three places.

---

* Trans. Am. Inst., Vol. XX, 3.

After finishing, the tanks are painted on the outside, staves and bottoms, with three coats of white lead.

Dimensions.—Formerly, the dimensions of lixiviation-tanks were taken quite small: Ore-tanks not larger than 12 feet diameter and 3 to 4 feet deep; precipitating-tanks, solution-sumps and storage-tanks of corresponding dimensions. In recent works, however, ore-tanks of 16 to 22 feet diameter and 8 to 10 feet depth; precipitating-tanks, solution-sumps and storage-tanks of 12 feet diameter and 8 to 10 feet depth are put up.* As can readily be seen, the care and attention required to finish a charge in an ore-tank, or to precipitate a solution in a precipitating-tank, are independent of the size of the vessel; hence, the great advantages of large sizes.

The capacity of an ore-tank for twenty-four hours depends upon the specific gravity of the ore, the quantity of first and second wash-water, and of stock-solutions required for treatment, but principally upon the rate of lixiviation. Capacity increases in proportion to diameter, but remains nearly stationary so far as depth is concerned; that is, the same number of ore-tanks will be required whether their depth is 10 feet or only 4 or 5 feet, in order to treat a stipulated quantity of ore per day. In fact, should the rate of lixiviation increase with reduced depth, the same number of shallow tanks would put through in twenty-four hours more ore than deep ones. The principal advantage of increased depth consists, therefore, only in reducing the number of charges treated.

---

* Tanks of much larger diameter are used for the cyanide process in South Africa, but I cannot see the advantage derived from excessive size, at least in treating silver-ores.

Sluicing Tailings.—Where water is abundant, tailings are now removed by sluicing, and great depth of the charge is no disadvantage. Even where water is scarce, and tailings have to be removed by hand, deep tanks should be used. It is only necessary to provide mechanical means for moving above the tanks large buckets into which the tailings are shovelled.

Filters for Ore-Tanks.—The false bottoms for the filter, and the latter itself, are prepared as follows: Wooden slats, 1¾ or 2 inches high and 1 inch wide, and separated 1 inch from each other, are fastened to the bottom by pins of hard wood. The inside of the slats, next to the bottom, is cut out in many places, ¾ or 1 inch deep and 3 inches wide, so that a free passage of the solution below the filter is established. Between the ends of the slats and the staves a clear space, 1½ inches wide, is left. A strip of wood, 1¾ or 2 inches high and 1 inch wide, previously cut with a saw in many places, and well soaked in water, so that it will bend easily, is now fastened round the slats, leaving an annular space, ½ inch wide, between the strip and the staves. One thickness of stiff matting, covering the slats and the circular strip, but not the annular space, forms the foundation of the filter-cloth proper. The latter, No. 10 canvas-duck, is cut to a diameter 6 inches greater than the inside of the tank, so that the ends can be pressed into the annular space described above, and kept in position by forcing down a ½-inch rope.

Sluice-Gates.—These are made of cast-iron with a discharge-opening 18 to 20 inches wide and 8 to 9 inches high. They are securely bolted to the staves of the tank so that the bottom of their discharge-opening is flush with the filter. The cast-iron door is covered with a sheet of rubber, and is held in place by an iron bar and wedge. It is convenient to have the door suspended by a counter-weight when removed. For very large ore-tanks, say 18 to 22 feet diameter, it is desirable to have two sluice-gates diametrically opposite to each other. A sluice-gate can also be placed in the center of the bottom of the tank. While the tailings are somewhat easier sluiced through a central opening, the construction of such a gate is more complicated, and it is more difficult of access. I have found that lateral sluice-gates answer the purpose perfectly.

Solution-Outlets for the Ore-Tanks.—These are made of 2-inch, six-ply, rubber steam-hose, which is inserted in the following way: A piece of clear plank, 3 or 4 inches thick, is fastened to the bottom of the ore-tank with wood-screws, and a hole, having the exact size of the outside diameter of the hose, is bored through the bottom and the plank, at a flat angle of about 30°. Through this hole the hose is forced. All joints are made with thick white lead, the wood-screws being also bedded in this material. A pin of hard wood is finally driven through the end of the hose into the wood. If the solution-hose is permanently connected with a Montejus (see § 1106) for producing a vacuum below the filter, and leaching with water from below the filter is desirable, a second hose should be inserted in the bottom of the tank and connected with the water-pipe. Finally, a third outlet may be provided and connected with a Koerting ejector (see § 1107), if circulation of extra-solution is necessary.

Launders for Solutions in front of Ore-Tanks.—These launders, 6 inches wide and 8 inches deep, are made of clear 1½-inch lumber and painted inside and outside with asphalt varnish. They are placed level, and should be held together with properly constructed braces. Connection with precipitating-tanks is made by 3-inch, six-ply rubber hose, which is inserted in the same way as the hose in the bottom of the ore-tanks, but the hole is not bored slanting.

Launders are needed as follows: one for silver-bearing wash-water, one for waste-water, one for solution, one for weak solution. If it is desirable to treat ore at different periods with cold and hot solutions, and to keep these solutions in rotation separately, a fifth launder must be added. Launders are either placed a short distance below the ore-tanks (from bottom of staves to top of launders, about 9 inches), and above the precipitating-tanks (about 6 inches from bottom of launders to top of precipitating-tank); or 5 to 6 feet above the top of precipitating-tanks, and below the upper floor round ore-tanks. The first position is taken if no artificial means are used to increase the rate of lixiviation; the second if the rate of lixiviation is increased by a Montejus with vacuum. (See § 1106.) The high position in the latter case is necessary to give sufficient head to the discharge through the 3-inch hose of the launder, when the solution is suddenly pressed up from the Montejus.

Boxes for Chemicals.—Solutions introduced to the ore-tanks are first conducted to a wooden box or barrel, with holes in the sides, standing on top of the charge. This prevents stirring up of the ore by a strong current of solution. These boxes also receive the copper sulphate and sodium hyposulphite for making extra-solution in the ore-tank.

Tail-Races.—If tailings are discharged by sluicing, proper attention should be paid to the construction of tail-races. Their inclination depends, of course, on the specific gravity and coarseness of the material; it should never be less than ¾-inch to the foot.

Precipitating-Tanks.

Stirring.—Stirring the solution by hand with an oar has now been entirely superseded by stirring with compressed air. This is done by holding a ¾-inch gas-pipe to the bottom of the precipitating-tank, the pipe being connected by a rubber hose with the supply-pipe for compressed air.

Decanting-Pipes.—The decanting of the clear solution, after precipitation, is done by a swinging 2-inch gas-pipe, entering the side of the tank near its bottom, through a stuffing-box. By lowering the elbow inside of the tank, the clear solution is gradually decanted.

Outlet for Precipitates.—Precipitates are discharged by a 2-inch asbestos-packed angle-cock, inserted close to the bottom of the tank through the staves.

Launders.—In front of the precipitating-tanks are launders, leading the decanted solution to the solution-sumps, and running decanted wash-water or weak solution to waste. If cold and hot solutions are kept in rotation separately, two launders must be placed in front of the silver precipitating-tanks, one leading to the sump for cold, the other to that for hot solution. The launders at the back, conveying precipitates to the storage-tanks for sulphides, wash-water precipitate and lead carbonate, should be inclined, with a fall of about 4 feet in 100 feet. These launders are constructed in the same way as those in front of the ore-tanks, but need not be so large.

Storage-Tanks for Precipitates.

These are made about 10 feet in diameter and 3 feet deep. A 2-inch rubber hose forms the outlet for precipitates through the bottom; it is connected with the press-tank. A 1-inch decanting-pipe for solution is inserted through the staves about 1 foot above the bottom.

Solution-Sumps and Storage-Tanks.

These should be made of the same diameter as the precipitating-tanks, but about 1 foot less in depth. Two of each kind should be provided, especially where solutions are heated, and hot and cold solutions are kept in rotation separately.

§ 1102. *Precipitating-Tanks for Wash-Water.*

If it is desirable to precipitate silver from the wash-water by copper or iron and sulphuric acid, the process is best conducted in a long, rectangular, lead-lined box, provided with many partitions through which the solution is made to circulate, up and down, by a Koerting ejector. The steam from the latter heats, at the same time, the solution, and precipitation of the silver is quickly accomplished.

§ 1103. *Sodium Sulphide Tanks.*

The sodium sulphide mixing-tank is made of cast-iron, 3 feet diameter and 7 feet deep. Its great depth is necessary because the concentrated lye foams considerably while adding the sulphur. The bottom is covered 1 inch deep with lead, to prevent its wearing out where steam is admitted for heating the lye. The two sodium sulphide storage-tanks, receiving the diluted solution, are made of $\frac{3}{16}$-inch boiler-iron, about 6 feet in diameter and 5 feet deep. All the tanks are provided with $1\frac{1}{2}$-inch asbestos-packed cocks. It may be desirable to increase the contents of sodium hyposulphite in the solution by rapid oxidation. For this purpose a gas-pipe coil, perforated by numerous small holes, is placed on the bottom of the tank and connected with the pipe for compressed air. The air is then forced in small bubbles through the solution.

§ 1104. *Apparatus for Heating the Stock-Solution.*

Apparatus for heating stock-solution can be placed either in the solution-sumps or in the storage-tanks, depending on the most convenient location of the boilers. Coils of lead pipe, through which steam is conducted, are very effective, but, unfortunately, not sufficiently durable to warrant their cost. Although lead is not attacked by a pure hyposulphite solution, the stock-solution contains sulphates and chlorides, and these may form lead sulphate and chloride, both soluble in hyposulphite salts. Besides, we have to consider that the stock-solution is never entirely free from silver and copper. The lead, after being in use for some time, becomes very brittle.

At the Marsac mill, heavy cast-iron pipes, one foot in diameter, have been substituted for lead-coils with satisfactory results regarding durability and cost.

I suggest giving to these heaters the shape of large, round, hollow disks, for a more economical utilization of steam.

Heaters should either be provided with steam-traps, or else the escaping condensed water and steam should be conducted to the feed-water tank for the boilers.

§ 1105. *Apparatus for Raising Stock-Solution to the Storage-Tanks.*

Plunger-Pumps.—Pumps of the Knowles type, as made and kept in stock by manufacturers, are not durable in contact with hyposulphite solutions, none of the metals or their alloys resisting its corroding action. To make such pumps durable, valves, valve-seats and plungers should be made of hard rubber or glass. It is difficult to induce manufacturers to take such orders except at extravagant prices. The same objections apply to centrifugal and rotary pumps. They can be used, however, to advantage in case the solution has to be elevated to inconsiderable height, for example, where precipitating-tanks for lead and silver are on the same level, and the decanted solution from the former has to be transferred to the latter tanks.

Pumps of the Koerting Type.—Such pumps must be made of material not affected by the solution. Their principal objection is the dilution of the solution by condensed steam.

The Montejus System.—A Montejus is an iron tank from which a solution is raised through a pipe by admitting steam or compressed air above the solution. (For protection of the inside, see § 1113.) With steam, the tank should be in the shape of a long cylinder of relatively small diameter, standing upright, to avoid condensation as much as possible. With compressed air as motive power, the shape of the tank is not material. It is then best to place it so that the axis of the cylinder is horizontal. One of the heads of the cylinder is put on with bolts and a gasket, so that the interior can be easily coated with an asphaltic compound.

These tanks have two openings: one at the bottom, through which the tank is filled and discharged by a three-way cock, connected with the pipes leading to the solution-sumps and the storage-tanks; one on the top, through which air escapes while the tank is filling, and steam or compressed air is introduced to elevate the solution, which changes are also effected by a three-way cock.

For works of large capacity, two Montejus are placed side by side below the solution-sumps, so that one is filling while the other is discharged. They should hold about 200 cubic feet. Convenient dimensions are: 4 feet diameter and 16 feet length, or 4½ feet diameter and 14 feet length, with solution-pipes from 3 to 4 inches diameter.

For large works, compressed air deserves decidedly the preference. The air-compressor should give about 30 pounds pressure per square inch.

§ 1106. *Apparatus for Increasing the Rate of Lixiviation.*

Suction-Hose.—The means formerly employed for increasing the rate of lixiviation have been to give the hose of the solution-outlet below the filter of the ore-tank great length, and discharge the solution at considerable depth below the tank, taking care that the hose remains filled with solution, and creates a suction by the weight of the solution-column. This method is objectionable, not only because it is difficult to get rid of all the air below the filter, but also for the reason that the solution has to be raised again to the precipitating-tanks, unless a very steep mill-site permits their position much below the ore-tanks.

Pumps.—These are not recommended for reasons given in § 1105.

Koerting Ejectors.—A Koerting ejector is effective provided it is not made of material liable to be corroded by the solution. The lead-lined ejectors, sold by A. Allen, New York, are not durable ; even the platinum steam-nozzles wear out. Ejectors of porcelain, with hard rubber steam-nozzles, are manufactured by the Koerting Brothers, Hanover, Germany. The objection to a Koerting ejector is, however, the dilution of the solution by condensed steam. The apparatus also acts intermittently, unless carefully regulated to meet the capacity of the filter.

The following precautions should be observed in using a Koerting ejector for increasing the rate of lixiviation : (1) The capacity of the ejector should correspond as nearly as possible with the work it has to perform. (2) The steam should be dry and of high pressure so that the dilution of the solution is reduced to a minimum. (3) The ejector should be placed in such a position that the suction inlet stands upward, and the hose coming from the ore-tank runs down without sagging.

Montejus.—The best effect is unquestionably produced by a Montejus. For this purpose the opening at the bottom is connected with a pipe through which, by a three-way cock, the solution either runs in from the hose at the bottom of the ore-tank, or is raised to the launders in communication with the precipitating-tanks. The outlet on top has a pipe with a three-way cock, through which compressed air can be introduced for raising the solution, or a suction is created by a Koerting vacuum-pump for filling the tank, or communication is shut off both from the air-compressor and vacuum-pump, provided the apparatus is idle. Only one Koerting vacuum-pump, producing a vacuum of 68 cm. quicksilver, is needed for a number of Montejus tanks, the latter being all in communication with a pipe, at the end of which the Koerting is placed.

These vacuum-pumps are best obtained from the Koerting Brothers, Hanover, Germany.

Necessarily, the operation of the apparatus is intermittent ; but the discharge of the Montejus takes such a short time that this is not detrimental.*

§ 1107. *Apparatus for Circulating Extra-Solution.*

In circulating extra-solution of standard composition, only a quantity is used sufficient to saturate the charge in the ore-tank, and this is allowed to filter repeatedly and continuously through the ore. As will readily be seen, the Montejus system cannot be used for this purpose. Here we must rely upon a Koerting ejector, made of porcelain.*

---

* This method has not yet been applied in practice.

§ 1108. *Filter-Press and Press-Tank.*

For lixiviation-mills of various capacities I recommend Johnson's filter-press, with 24 to 36 chambers, 15 to 18 inches diameter, without distance-rings and with closed delivery.

If it is desirable to free the precipitates from adhering stock-solution, which is the case if sulphides are refined by a humid process in the mill, a filter-press should be used in which the cakes can be washed after the chambers have been filled.

The chambers are of cast-iron, with a coating of asphalt varnish, which should be occasionally renewed.

The proper way of charging the filter-press is by a press-tank with compressed air, which should be safe for 150 pounds pressure to the square inch.

The press-tank is provided with a conical bottom so that its contents can be completely discharged into the filter-press. An opening with cock in the side, close to the top, connects with the discharge-hose of a storage-tank for precipitates; and a pipe, with a three-way cock, through which compressed air can be admitted or discharged, perforates the top. A separate press-tank and filter-press should be provided for each class of precipitates.

Since, in working with high pressure, the solutions leaving the filter-press are not always perfectly clear, they should be returned to their respective precipitating-tanks.

Even with closed delivery a small sump is required below the filter-press, because the filter-joints between the chambers leak sometimes considerably. From these sumps the turbid solutions are returned by a Koerting ejector to their respective precipitating-tanks.

§ 1109. *Drying-Chamber for Precipitates.*

The cakes of precipitates, coming from the filter-press, are placed into sheet-iron pans or trays and put into a drying-chamber. The latter is heated either by steam-pipes placed between the rows of pans, or by a hot-air stove. The necessary draught is produced by a Koerting exhaust-blower which removes the moisture-laden air. Care has to be taken that the temperature of the chamber does not rise to a point at which the free sulphur in sulphide precipitates ignites. (A description and drawing of a drying-chamber for sulphides is given in § 1114.)

---

* The same rules, given in § 1106, apply here in regard to position and connection of the ejector.

§ 1110. *Auxiliary Machinery.*

Air-Compressors.—For Montejus and for the press-tanks, two separate air-compressors are needed, the one furnishing air of about 30 pounds, the other of 150 pounds pressure. The size of the compressors must be calculated to meet the demands of the capacity of the works. The Montejus air-compressor should have a capacity to raise all the stock-solution needed per day in about 8 hours, so that the plant is not in operation all the time and the solution can be transferred quickly.

Sulphide-Sampler.—The breaking-up of the lumps of sulphides and other precipitates, as they come from the drying-chamber, is done in an ordinary small sample-grinder, as sold by any western foundry.

Boilers.—If considerable steam is needed for heating the stock-solution and the wash-water prior to its precipitation by scrap-iron or copper; also for running Koerting ejectors and vacuum-pumps, a separate boiler is desirable for the lixiviation-plant proper.

Fire-Pumps for Sluicing Tailings.—If the water in the pipe-line has not sufficient head for sluicing tailings, a fire-pump for this purpose becomes necessary. A steam-pump of the Knowles type may be used with 10-inch steam- and 5-inch water-cylinder, 12-inch stroke and 4-inch suction and discharge. I also recommend a Koerting steam-jet fire-pump, which is much cheaper than a plunger-pump and not less efficient.

§ 1111. *Pipe-Lines.*

Iron gas-pipes are used for conveying stock-solution and sodium sulphide solution.

They should be joined by flange-couplings. The size of the stock-solution pipes above the ore-tanks depends on the capacity of the works, and the position of the storage-tanks, producing more or less head. In large works they are generally taken from 3 to 4 inches diameter.

§ 1112. *Cocks.*

The only cocks resisting for a long time the corroding action of lixiviation-solutions are the Pratt-Cady iron-body, Bower-Barffed plugs, asbestos-packed cocks, sold by Fairbanks & Co., New York. These cocks last very well, even on the sodium sulphide mixing-tank. They were first introduced in lixiviation-works by the writer at the Marsac mill, Park City, Utah.

§ 1113. *Protection of Iron by Asphaltic Covering, and other Notes on Construction.*

Castings and Pipes.—Castings and pipes exposed to hyposulphite solutions must be protected by an asphaltic covering; the pipes, of course, on the inside. Pipes conveying sodium sulphide solution do not require such covering. A pure asphalt, called "Maltha," manufactured in San Francisco, the residue from distillation of heavy California mineral oils, now shipped in considerable quantities to the eastern States, is exceedingly well adapted to this purpose. The maltha is melted in a sheet-iron trough, and the clean and dry castings or pipes are immersed in the bath. They should remain there until the iron has reached the temperature of the liquid asphalt; this insures firm adhesion of the asphaltic covering to the iron.

Montejus tanks and the press-tanks need different treatment. Here a malthine varnish is used. To the clean and dry inside of the vessels, a coat of varnish is applied. After drying, a second coat is put on, and, as the painting progresses, cotton cloth—weight about 6 ounces per yard—is pasted on, care being taken to saturate overlapping joints with varnish. On top of the tank the cloth is drawn over the flange. The cover is treated in the same way. Finally, a third coat of varnish is put over the cotton cloth. After perfect drying, the cover is bolted on, using a soft rope saturated with varnish for a gasket.

Fastening Castings and Flanges to Tanks by Bolts.—Bore holes to fit bolts closely; countersink bolt-heads into wood and cover with thick white lead. Washers behind bolt-heads are also bedded in white lead. Bolts are first covered with asphalt.

Connecting Rubber Hose with Gas-Pipe Nipples.—The following method was communicated to me by F. M. Johnson: Draw out the nipple to fit the inside of the hose closely; heat the nipple so that it will melt rubber without burning it; then press the nipple into the hose about 4 inches and immerse immediately in asphalt varnish for cooling. The hose adheres so firmly to the iron that it can only be removed by cutting.

§ 1114. *General Remarks about Construction.*

The apparatus used in lixiviation, and fully described above, is all so simple that I have considered it superfluous to furnish detailed drawing of the same. So far as the arrangement of the plant is concerned, I must say that where separate precipitation of lead is practiced, it is more convenient to place the precipitating-tanks for lead on the same level with those for silver, and not above the latter, and elevate the solutions from the former to the latter by a Koerting ejector, or by a centrifugal pump. This arrangement reduces the expense of grading for the mill-site, and the building becomes more compact.

How many ore-tanks and precipitating-tanks a lixiviation-mill of stated capacity should have cannot be stated in general terms. The number of tons that can be treated in a plant of given size depends, so far as the ore-tanks are concerned, on the rate of lixiviation, and on the volume of stock-solution required per ton of ore. The number of precipitating-tanks is governed only by the volume of solution coming to precipitation; but there can never be less than two precipitating-tanks for each kind of solution, namely, for precipitating wash-water, for lead, for silver, and for weak solution. Wash-water and weak solution are precipitated together in some mills, but this is not good practice. Hence, before making working-plans for a lixiviation-mill, the ore to be treated should be thoroughly tested in an experimental plant. Such a plant exists at the Ontario mill, Park City, Utah, and tests are made there upon application.

The accompanying plate shows in detail the ground-plan and elevation of a modern lixiviation-plant designed for ore not requiring separate precipitation of lead. Where this is necessary, three more precipitating-tanks for lead, a centrifugal or Koerting pump, and a storage-tank for lead carbonate, with press-tank and filter-press, should be added.

In another plate a drying-chamber for sulphides, with hot-air stove, is shown. The cold air enters the top of the stove and moves down through a spiral passage. Before it enters the drying-chamber, it passes through a brick-chamber which serves as a store-room and equalizer of heat. The draught is produced by a Koerting ventilator. Both ends of the drying-chamber are closed by double, sheet-iron doors, not shown in the drawing. The outside of the hot-air stove should be covered with a non-conductor.

§ 1115. *Cost of Modern Lixiviation-Plants.*

Plant of 60 tons daily capacity at the Blue Bird mill, Butte, Montana.—Below is a statement of the actual cost of the Blue Bird lixiviation-plant, according to C. A. Hoyt:

Specification of Plant.

4 Ore-tanks, 17 ft. diam.; 9 ft. deep.

8 Precipitating-tanks, 9 ft. diam.; 9 ft. deep.

3 Solution storage-tanks, 9 ft. diam.; 9 ft. deep.

1 Stock-solution sump; 3 Storage-tanks for precipitates; each 20 ft. by 3 ft. 6 in., and 3 ft. 6 in. deep.

1 Cast-iron sodium sulphide mixing-tank, 3 ft. diam.; 7 ft. deep.

1 Wrought-iron sodium sulphide storage-tank, 4 ft. diam.; 4 ft. deep.

1 Wrought-iron sodium carbonate storage-tank, same dimensions as above.

Filter-presses, pumps, etc.

The building covers an area of 5,000 square feet.

Total Cost of Plant, including Grading and Building: $20,014.00.

Of this $3,039.00 was for lumber, and $7,265.00 for labor.

Plant of 200 to 250 tons daily capacity for tailings, or 150 tons for ore, at the Ontario mill, Park City, Utah.—Only the building of this plant is finished, but most of the materials and machinery are on the ground. The estimate to complete construction is furnished by Salkeld & Behr.

Specification.	Diameter. Feet.	Depth. Feet.
10 Ore-tanks	17	9
14 Precipitating-tanks	12	9
6 Storage-tanks for precipitates	10	4
2 Solution-sumps, 2 Solution storage-tanks,	12	8
2 Montejus	4	16
2 Sodium sulphide mixing-tanks	3	7
2 Sodium sulphide storage-tanks	6	5

The rest of the specification is seen from the statement of cost:

Cost of Building.

Grading		$2,831.75
Rock-work: Labor	$1,555.25	
Rock	1,885.79	
Sand	529.47	
Lime	399.82	4,370.33
Carpenter-work: Labor	3,641.26	
166,680 ft. Oregon lumber	3,333.60	
183,810 ft. Native lumber	3,807.85	
Hauling Oregon lumber	262.33	
225,000 Shingles	573.75	
91 Windows	287.48	11,906.27
Sundries: Teams	127.50	
Nails, rope, etc.	647.27	
Machine-shop work	145.60	
Supplies and work at Ontario mill	561.62	1,481.99
Plans and supervision	3,101.00	3,101.00
Total		$23,691.34

Cost of Materials and Machinery on Hand.

Lumber for tanks	$2,752.00	
False bottom strips	340.00	
7,742 ft. 1½ in. sugar pine	402.84	
Freight and hauling	1,273.78	$4,768.62
Tank-bands and lugs	1,238.35	
Freight and hauling	431.42	1,669.77
Machinery:		
One Ingersol-Sargent air-compressor, 8″ × 10″ × 10″	735.00	
One Ingersol-Sargent air-compressor, 8″ × 5″ × 10″	700.00	
One centrifugal pump	100.00	
24 pipe-coils	575.00	
Line-shafting, pulleys	159.90	
2 Montejus; 3 press-tanks; sodium sulphide mixing and storage-tanks; 10 sluice-gates; sulphide drier with 86 pans	3,814.00	
Freight and hauling	1,073.49	7,157.39
Total		$13,595.78

Estimate to Complete the Plant.

3 Filter-presses with duplicate set of filters, etc.	$1,500.00
Pipes, fittings, cocks, valves, hose	2,550.00
36,000 pounds cement for floors	450.00
Sample-grinder and sampling-machine	250.00
Stone-foundation below tanks	1,800.00
Labor and supervision	10,220.00
Total	$16,770.00

Recapitulation.

Building	$23,691.34
Materials and machinery on hand	13,595.78
Estimate to complete	16,770.00
Grand Total	$54,057.12

CHAPTER XII.

THE SOLUTIONS.

§ 1201. *Definition of Terms.*

An "*acid ore*" is a roasted ore producing a first wash-water which gives an acid reaction.

An "*alkaline ore*" is a roasted ore, containing caustic lime, which produces a first wash-water giving an alkaline reaction.

An "*arsenical alkaline ore*" differs from the last named one in that it contains an unusual amount of arsenates. The only example of this kind, so far observed, is the Yedras ore.

"*Circulating*" a solution means withdrawing it by means of a Koerting ejector from below the filter of an ore-tank, and returning the solution to the latter, so that it filters continuously and repeatedly through the ore.

"*The volume of solution to saturate a charge*" is the quantity of liquid that fills the space between the particles of ore. If a charge of ore is already saturated with one liquid, and the latter is displaced by another, the volume required to saturate the charge is reduced from 6 to 10 per cent., because some of the former liquid remains.

§ 1202. *The Ordinary or Stock-Solution.*

Preparation.—The ordinary or stock-solution is made by dissolving a proper quantity of sodium hyposulphite in a measured quantity of water in the storage-tanks. As the weight of a cubic foot of water is 62½ pounds, this number of pounds of sodium hyposulphite, added to each 100 cubic feet of water, will produce a solution of one per cent. concentration, and so on.

Concentration in Sodium Hyposulphite.—Practical experience has demonstrated that it is, in most cases, not economical to work with a solution of higher concentration than 2 per cent., even if high-grade ores are treated. In that case, increased volume of the solution is made to do the work. With low-grade ores, a solution of 1 per cent., and one of less concentration, may be most profitable.

In starting up new works, it is best to begin with a 1½ per cent. solution, and then diminish or increase its concentration according to circumstances.

Volume of Stock-Solution used per Ton of Ore.—The volume of ordinary or stock-solution used per ton of ore may vary between 70 and 225 cubic feet.

Change in Concentration by Continued Use.—The strength of the solution, after continued use, changes, and in most cases it becomes weaker.

This subject has already been fully discussed in § 608.

Loss in Sodium Hyposulphite.—The actual loss in sodium hyposulphite, treating roasted ores by the Russell process, is from ¾ to 7 pounds per ton of ore, and rarely exceeds the latter limit. An unusually high consumption of hyposulphite, of nearly 10 pounds per ton of ore, is recorded at the Aspen mill, Colorado. Precipitation was done, in part, with commercial Na_2S, which does not contain hyposulphite as is the case with sodium sulphide prepared in the mill.

A considerable difference exists between "acid" and "alkaline" ores. It is less with alkaline ores, because, as will be seen in the next paragraph, less copper sulphate is used for making the extra-solution. Besides, cuprous chloride often exists in acid roasted ores, which is not all extracted by the first wash-water and is dissolved by the ordinary solution first applied. In this way a weak extra-solution is formed, which deteriorates quicker than an ordinary solution without copper.

Gain in Sodium Hyposulphite.—The solution gains in strength after precipitation with sodium or calcium sulphide. The more silver, lead, and especially copper precipitated, the larger will be the quantity of hyposulphite salts added to the solution, so that cases may exist where all the loss in hyposulphite salts is not only compensated, but the solution even increases in strength. This may occur if base ores, after chloridizing-roasting, are treated by ordinary lixiviation. Where extra-solution is used, the stock-solution can never gain strength in hyposulphite salts.

Maintenance in Strength.—After the most advantageous concentration of the solution has been determined by experience, it should be maintained by constant addition of sodium hyposulphite. This is done—if the Russell process is used—in making the extra-solution, as will be shown in the next paragraph.

Concentration in other Salts.—The original stock-solution is subjected to other changes, outside of those mentioned above, in consequence of a gradual accumulation of sodium chloride, sulphate, and tetrathionate. It can easily be seen whence these salts are derived. Sodium chloride and tetrathionate do absolutely no harm. Sodium sulphate diminishes the solvent energy of the solution only slightly. Calcium-salts are introduced if gypsum is present in raw ore, or if caustic lime or calcium sulphate exist in roasted ore, or if calcium sulphide is used as precipitant for silver. In case the lead is precipitated by soda, the calcium is also precipitated, and no calcium-salts can exist in the solution.

The concentration of the solution in these salts, however, does not go on indefinitely, but reaches a maximum and then remains stationary. This effect is produced by the wash-water, which either precedes and follows, or merely follows the lixiviation-solution. In this operation a dilution of the lixiviation-solution takes place.

That sodium tetrathionate is again converted into hyposulphite, has already been stated in § 605.

Caustic Stock-Solution.—There is, however, another change in the condition of the stock-solution that is of great importance, namely, its assuming a caustic reaction. How seriously this condition of the solution may affect the final result of lixiviation has been shown in § 218. Whenever the stock-solution shows the presence of caustics, to an injurious extent, it must be neutralized with dilute sulphuric acid. The latter is added in the silver precipitating-tanks, immediately after the precipitation of the sulphides has been finished, and before the solution is decanted and returned to the storage-tanks. The reaction of the solution on litmus-paper should be closely observed so that not more acid is consumed than is absolutely necessary. In using the Russell process, a caustic stock-solution is neutralized by copper sulphate, but this is done at the expense of that salt.

Volume of the Stock-Solution in the Mill.—It is evident that the amount of stock-solution, kept on hand and in circulation, should be reduced to a minimum. The quantity of stock-solution required for works of given daily capacity depends upon the time of lixiviating with extra- and ordinary solution, and upon the volume of solution necessary to saturate a ton of ore. This will be made clear by an example. Suppose the daily capacity is 80 tons; one ton of ore needs 9 cub. ft. of solution for saturation; the time of lixiviating with extra-solution is 8 hours, and with ordinary solution 36 hours. In that case, 146 tons of ore are constantly under treatment, and they require 1314 cub. ft. of solution for saturation; besides, a certain quantity of solution stands above the charge in the tanks, say 260 cub. ft.; one precipitating-tank is full and settling, and a second one is half-filled, say 1200 cub. ft. for both; finally, we must leave sufficient solution in the storage-tanks to supply the ore-tanks, say 800 cub. ft. This would give a total of at least 3574 cub. ft.

Temperature of the Stock-Solution.—In most cases the best results are obtained in lixiviating with an ordinary solution heated from 35° to 50° C.

§ 1203. *The Extra-Solution.*

Extra-Solution not Kept in Stock.—The extra-solution is not kept in stock, but made in charges as needed, immediately before use, for reasons fully stated in § 317.

Composition of Extra-Solution.—Extra-solution of standard composition is used for raw, acid, and alkaline arsenical ores, while alkaline ores are in most cases treated by extra-solution with much less copper than is required for standard composition. In the latter case, the proportion of $CuSO_4+5aq.$, to $Na_2S_2O_3+5aq.$, varies between 1 to 3, or 1 to 4, and even extra-solutions much weaker in copper are profitably used, as will be seen later.

Rules for Preparing Standard Extra-Solution.—The stock-solution, used in the preparation of standard extra-solution, is generally of such low concentration in sodium hyposulphite, that it does not contain a sufficient quantity of this salt to form a solution of desired concentration in copper. Hence, in dissolving the copper sulphate, a proper amount of sodium hyposulphite must be added. I will illustrate this by an example. We will suppose that ore is treated which requires 8 cub. ft. of standard extra-solution per ton; that the normal concentration of the stock-solution is 1½ per cent. in sodium hyposulphite; that 5 pounds of copper sulphate are needed per ton of ore. The 8 cub. ft. of stock-solution would represent only 7½ pounds of sodium hyposulphite; hence, not less than 3¾ pounds of this salt should be added for each 5 pounds of copper sulphate in 8 cub. ft. of solution, in order to produce a standard extra-solution.

Before making standard extra-solution it should be ascertained whether the stock-solution is alkaline or acid. If alkaline, an extra-solution will result which deteriorates rapidly, as shown in § 311. Commercial copper sulphate generally contains sufficient free acid to produce a permanent standard extra-solution from a neutral stock-solution. Extra-solutions with twice the amount of sodium hyposulphite required for standard composition are quite permanent, if neutral.

Modes of Preparation: First Method.—Extra-solution is either prepared in a tank separately, or on the charge itself in the ore-tank. The first method is used if raw ores are treated first with extra-solution, no other liquid having preceded. The necessary amounts of copper sulphate, together with sodium hyposulphite, put into a box with holes, placed above the extra-solution tank, are dissolved by ordinary solution. Before this is done, however, the tank is filled with about two-thirds of the required amount of ordinary solution. In this way the strong copper-solution meets at once a sufficient amount of sodium hyposulphite to prevent precipitation of the canary-yellow salt of Lenz. If the extra-solution is to be used warm, it is best to heat the stock-solution to the required temperature before turning it into the extra-tank.

Testing Standard Extra-Solution.—A sample of standard extra-solution should be clear, and not become turbid if shaken with air in a partly filled flask. Turbidity would indicate the entire absence of free acid. I have shown, in Chapter III., III., that acid reaction with litmus-paper is by no means a proof that the extra-solution contains free acid. Hence, the litmus-paper test cannot be relied upon. If the stock-solution, used in preparing the extra-solution, was not clear originally, the test described above should be made with a few c. c. of extra-solution after filtering. If the standard extra-solution turns dark when heated to 65° C., this would indicate the presence of too much free acid.

The Second Method.—The second method of preparing extra-solution is used if the latter is preceded by wash-water or ordinary solution. The discharge-pipe of the ore-tank is closed as soon as the liquid has sunk to the surface of the ore. Proper amounts of copper sulphate and sodium hyposulphite, placed in the perforated box above the tank, are dissolved by cold or warm stock-solution, which is added until the space above the ore, say 12 inches deep, is filled. The extra-solution should be well stirred before it is allowed to sink through the charge. As this volume is insufficient for a deep charge in the tank, a new quantity of extra-solution is made as soon as the first one has been allowed to sink to the surface of the ore, etc., until the extra-solution has displaced all the liquid previously contained in the ore, when circulation is commenced, provided the character of the ore calls for such treatment.

Percentage of Copper Sulphate in Extra-Solution.—The percentage of copper sulphate in standard extra-solution for raw, acid, and arsenical alkaline ores varies between 0.6 and 1.2 per cent.; that of extra-solution for alkaline ores, in most cases, between 0.1 and 0.33 per cent.

Volume.—The volume of standard extra-solution used for raw, acid, and arsenical alkaline ores is just sufficient to saturate the charge. It varies, according to specific gravity and fineness of pulverizing, for raw ores between 6½ to 9 cubic feet; for acid and arsenical alkaline ores, between 10 to 14 cubic feet per ton of ore. These small quantities, as compared with the volume of ordinary solution, are entirely sufficient to accomplish the object in view.

For alkaline ores, the quantity of extra-solution is from 4 to 6 times the volume to saturate the charge.

Depth.—The quantity of extra-solution required can also be expressed in percentage of the depth of the wet charge, that is, its depth after lixiviation has commenced. It is from 36 to 42 per cent. for raw ores; from 34 to 46 per cent. for acid and arsenical alkaline ores; and from 150 to 300 per cent. for alkaline ores.

Circulation.—In using standard extra-solution on raw and acid ores, it is generally circulated by a Koerting ejector. Thus, the small volume of this solution is repeatedly brought in contact with all the ore-particles, producing the best effect obtainable. A solution in motion is a much better solvent than one at rest. Still, there are exceptions to this rule—as will be seen in § 1407—quite difficult to explain.

The extra-solution on alkaline ores is not circulated, but manipulated like ordinary solution.

Sometimes it is profitable to transfer the extra-solution used on one charge to a charge in another ore-tank.

Effect of one Pound Copper Sulphate.—From practical experience it appears that in treating acid ores, 1 pound copper sulphate, consumed in standard extra-solution, extracts— after the ordinary solution has done its work—from 1.68 to 1.81 ounces silver per ton of ore. With alkaline ores, the effect may be greater.

Temperature.—Extra-solution is either used cold or heated from 35° to 50° C.

§ 1204. *The Sodium Carbonate Solution.*

If ordinary soda-ash is used, it is dissolved in stock-solution and purified with standard extra-solution, which is added while stirring until a black precipitate ceases to appear. (See § 220.) The Solvay soda does not require purification after it has been dissolved in stock-solution.

§ 1205. *Caustic Lime.*

Caustic lime is slacked, and from the powder milk of lime is made with stock-solution in a bucket; or the solution in the precipitating-tank itself can be used for this purpose.

§ 1206. *The Sodium and Calcium Sulphide Solutions.*

The necessary instructions for their preparation have already been given in § 601.

CHAPTER XIII.

THE CHARGING AND DISCHARGING OF THE ORE-TANKS.—GENERAL RULES FOR HANDLING THE LIXIVIATION-SOLUTION.

§ 1301. *Charging the Ore-Tanks.*

Charging by Cars.—The charging of the tanks should be done by cars. Arrangements with hoppers above the tanks, including elevators and conveyors, are not economical. There is, in every case, a limit to the economy of replacing manual labor by machinery. The best arrangement is to run a double car-track over each tank from the cooling-floor, and to use dumping-cars of proper construction.

Before commencing the charging of roasted ore, water should be run into the tank to cover the filter, which otherwise might be injured by some hot lumps of ore; or the tank is filled with sufficient water to saturate the charge.

Charge the Ore Dry.—The ore must be charged dry. Roasted ores, of course, contain a certain amount of moisture if wetted down on the cooling-floor; but this is not what is meant by charging wet. The effect of charging ore wet will be described and criticised in Chapter XV., on trough-lixiviation.

Charge the Ore Cold.—Roasted ore should not be charged hot, —that is at a temperature of 65° to 120° C.,—but cold,—that is below 50° C.,—if it is desirable to reduce the amount of silver taken up by the first wash-water to a minimum. If the ore contains caustic lime and is dumped hot into water, the "going back" of the chlorination is thereby considerably increased.

Weight of the Charge.—The weight of the charge should not be roughly estimated, but ascertained as accurately as possible. With dry raw ore this is effected easily enough; it is more difficult, however, with wetted down roasted ore, when the moisture enters into calculation. In the latter case, the samples taken from each car should be kept in a box with tight cover, so that moisture is not reduced by evaporation. To conduct lixiviation in a rational manner, reliable statistics must be kept. A correct basis should exist on which the "*apparent extraction*" of the silver, that is the percentage calculated from the values of the roasted ore and the tailings, can be compared with the "*actual extraction*," or the percentage of silver actually obtained in sulphides, including the silver extracted from the first wash-water, by a clean-up in the mill.

If a notable discrepancy between the two figures exists, it should be investigated. Loss in silver may occur in many ways: by leakage of solution; by inadvertence in running silver-bearing solution to waste; by wash-water; and by stealing of sulphides, which has to be guarded against especially in Mexico. The losses in silver before the ore comes to lixiviation do not concern us here.

Filling the Ore-Tanks.—The tanks are filled to such an extent that the top of the charge, after the solutions have been applied, is about 12 inches below the upper rim of the tank, for reasons that will become apparent in the next paragraph.

Shrinkage of the Charge.—The ore, whether raw or roasted, shrinks as soon as lixiviation-solution is applied to raw ores, or the first wash-water to roasted ores. This shrinkage is from 10 to 18 per cent. for raw, and from 12 to 24 per cent. for roasted ores. It is greater in the latter case on account of the removal of soluble salts. A few trials will establish how high a tank must be filled to leave the required empty space above the ore, after lixiviation has commenced. A deep tank, if the ore shrinks considerably, may be filled to the top.

Precautions.—After the tank is filled, the charge is levelled with a hoe. In doing this, the laborer should not step into it, or forcibly press down the ore. Such irregularities interfere with the uniform filtering of the solutions through the charge.

§ 1302. *General Rules for Handling the Lixiviation-Solutions.*

Before entering into a detailed description of the modifications in lixiviation necessary in special cases, it is best to establish general rules that must be followed under all circumstances, and without which the best results, and the greatest economy, cannot be reached.

Volume to Saturate.—The first step to be taken is to ascertain how many cubic feet of solution, or inches in depth, are required to saturate one ton of ore or a charge. In § 1203, it has already been noted between which limits these quantities vary for different classes of ore.

First Wash-Water.—All roasted ores are first treated with water, called the first wash-water. Whether the first wash-water is followed by ordinary or by extra-solution, it is imperative, at the end of this operation, to stop the discharge of the liquor as soon as the water has sunk to the surface of the ore. The space in the tank above the ore, say 12 inches deep, is now filled with solution, and the outlet kept open until the solution has sunk to the surface of the ore. This operation is repeated until all the wash-water is pressed out, and displaced by lixiviation-solution.

Lixiviation.—Now, the discharge of the liquor is turned into the launder leading to the precipitating-tanks. It is, of course, necessary to observe carefully when this change takes place, so that hyposulphite solution is not lost. Samples of the liquor must be tested frequently with sodium sulphide.

Extra-Solution.—In applying standard extra-solution to roasted or raw ore, after treatment with ordinary solution, and in the minimum quantity, namely, just sufficient to saturate the charge, the same *modus operandi* as described above is adopted, in order to displace the ordinary solution. As soon as this is accomplished, the discharge of the solution from the ore-tank is stopped, and circulation of the standard extra-solution is commenced, or it is left standing in the charge.

Second Wash-Water.—At the end of lixiviating with hyposulphite solutions, the second wash-water follows. Here again the discharge of the solution is stopped as soon as it sinks to the surface of the ore. Several charges of water are now turned on, sufficient to displace the lixiviation-solution, when discharging into the precipitating-tanks is stopped, and the wash-water is allowed to drain off, in case the tailings are shovelled out, or remains, if they are discharged by sluicing.

Measurement of all Solutions.—But outside of these measurements which are absolutely necessary, it is recommended to measure all the wash-water and lixiviation-solution consumed, so that only a fixed quantity, established by experience, is passed through the charge. This is most conveniently done by observing frequently the rate of lixiviation.

It can readily be seen that turning on solutions at random would only create confusion. It is not feasible to measure volumes either in the storage-tanks, or in the precipitating-tanks, since the former supply a number of ore-tanks, while the precipitating-tanks collect solution from several sources simultaneously. To use the last-mentioned system of measurement, it would be necessary to have for each ore-tank separate storage- or precipitating-tanks, which is not practicable.

The Rate of Lixiviation.

The rate of lixiviation is the number of inches in depth of a liquid which will pass through a charge of ore in a tank per hour. In other words, it is the facility with which a solution filters or sinks through a charge. It is important to know upon what circumstances the rate of lixiviation depends, especially since metallurgical literature is full of erroneous statements on this subject. We have to consider the depth of the charge, the fineness of crushing, the temperature of the solution, and the character of the ore.

Influence of Depth.—The depth of the charge, in contradiction to old notions, does not influence the rate of lixiviation so much as might be assumed at first sight, because the slight increase of compactness at the bottom of the tank, due to increased pressure, is more or less compensated by the greater "*head*" of the column of solution.

At the Curihuiriachic mill, according to Ellsworth Daggett, the rate of lixiviation for a charge 6 feet deep was even slightly greater than for one only 2 feet deep. This, however, is rarely the case.

Influence of Fine Crushing.—For raw ores, the rate of lixiviation diminishes with the fineness of crushing. Difficulties are especially experienced in treating tailings from raw amalgamation, if they have been ground repeatedly in pans. I cite the Bremen tailings, Silver City, New Mexico, as an example.

The average fineness of that part which could be successfully treated was such that 87.8 per cent. would pass through a No. 150 wire-screen. With this fineness of material, a vacuum of 14 inches of mercury produced a lixiviation-rate of about one-half inch. For roasted ores, the fineness of crushing has often no appreciable effect on the rate of lixiviation, if the dust is well chloridized, and charged dry into the ore-tanks. For instance, when treating

Ontario ore, a charge of the finest dust from the last dust-chambers of the Stetefeldt furnace lixiviated almost as rapidly as a charge of the coarse material from the shaft. This is due to the perfectly roasted condition of the dust, which causes the charge to maintain a soft and half-floating condition during the whole of the process, unless it is allowed to drain and settle. On the other hand, at Lake Valley, with a White furnace without an auxiliary fire for roasting the dust, the latter, even when mixed with the coarsest ore, was a great hindrance to rapid filtration.

For ore roasted in a Stetefeldt furnace, there is practically no limit to the fineness of crushing allowable.

Temperature.—The rate of lixiviation is greater with warm than with cold solutions.

Character of the Ore.—It is evident that in treating raw ores, the rate of lixiviation should be influenced by the character of the gangue, for example, if the latter contains clay or talcose matter; but why a considerable difference should exist between well roasted ores of almost the same composition is difficult to explain. A case of this kind is presented by the ores from the Daly and Ontario mines, both located on the same vein. The roasted Daly ore shows one of the lowest rates of lixiviation on record; it rarely exceeds, even with warm solutions, 3 to 4 inches. Ontario ore, on the contrary, lixiviates freely with a rate of 8 inches and above.

Limits of the Rate of Lixiviation.—The rate of lixiviation may vary between ½ and 12 inches, and may even exceed the latter figure.

Increasing the Rate of Lixiviation.—The rate of lixiviation can be greatly increased by creating a vacuum below the filter of the ore-tank. How this is accomplished has been stated in § 1106. If the natural rate of lixiviation is only a few inches, the Koerting ejector will generally double it.

Irregular Shrinkage of the Charge.—It occurs during lixiviation that, in consequence of irregular shrinkage, cracks form, either through the body of the ore, or the latter draws away from the sides of the tank. These defects should be at once corrected, especially in lixiviating with standard extra-solution. Otherwise, the solutions, following the way of least resistance, will do their work very ineffectually.

§ 1303. *Discharging of the Tailings and their Sampling.*

The removal of tailings by sluicing is by far the cheaper method, and requires from 12 to 18 cubic feet of water per ton of ore. If water is not abundant, that part of the first wash-water which runs to waste can be collected and pumped into a tank especially provided for that purpose. The sluicing is done with a hose, after opening the sluice-gate. Sluicing permits the use of deep charges, which are inconvenient for shovelling.

*Silver Sulphide on the Surface of the Charge.**—It may occur that the stock-solution contains extremely fine particles of silver sulphide in suspension, due to imperfect settling of the precipitates. In this case, the silver sulphide is arrested in a stratum on top of the charge. This portion of the tailings may be of higher value in silver than the original ore. It is carefully removed with a hoe. In some mills it is customary to return these tailings to the roasting-furnaces. Much cheaper and simpler is the method of treatment with extra-solution which extracts the silver rapidly and completely. A special tank may be used, or these tailings may be transferred to one of the ore-tanks before the charge is treated with extra-solution. If, in lixiviation, extra-solution were used last, these rich tailings could not be formed. But, as will be seen shortly, the process is always ended with ordinary, and not with extra-solution.

Sampling the Tailings.—At the Marsac mill, for instance, four different samples are taken: The first is taken, before the charge is sluiced out, by means of a long brass tube, 1½ inches in diameter, run vertically down through the charge in various places from top to bottom, whereby cores of tailings are obtained. The average of these cores constitutes the average tube-sample. After one-half of the charge has been sluiced out, leaving a vertical face across the center of the tank, the sampler gets into the tank and takes three samples as follows: (1) across the face about 1 foot below the horizontal surface of the tailings; (2) across the face about 1 foot from the bottom; (3) an average sample, consisting of one set across the top, one across the bottom, and one across the middle of the face, this mixture being called the average regular sample. Nos. 1 and 2 are merely for the information of the men in charge of lixiviation; their work being considered well done if the assay of the bottom tailings does not differ materially from that of the top. The assays of the average tube-sample, and of the average regular sample are reported in the metallurgical statistics of the mill. The average of these two samples for nearly two years, showed a difference of only 0.03 of an ounce of silver per ton.

---

* Sometimes a brown precipitate appears on the surface of the charge after extra-solution has been applied. This is copper sulphide, resulting from decomposition of extra-solution.

A notably higher value of the tailings near the filter shows that lixiviation was incomplete, that either the ordinary or the extra-solution did its work imperfectly, and that either a larger volume of ordinary solution, or else extra-solution of higher concentration in copper should be used. Higher tailings near the bottom of the tank may also result from a partial clogging of the filter, or of the passages below it, whereby channels of different resistance are formed. In such cases the filter should be taken up immediately.

It may occur, however, that high bottom tailings are produced in deep charges by the presence of caustic lime in the ore, the deleterious effect of which has not been counteracted by chemicals in the first wash-water. In this instance the damage done cannot be repaired by the application of larger volumes or more concentrated extra-solution.

CHAPTER XIV.

THE TREATMENT OF ROASTED AND RAW ORES.

All roasted ores require leaching with water, prior to the application of lixiviation-solutions, for reasons pointed out in § 502.

§ 1401. *The First Wash-Water.*

After the surface of the ore in the tank has been levelled, water is turned on, either on top of the charge, or from below the filter.

Leaching from Above.—This method is used, if the ore is of such a character that little silver is extracted by the first wash-water, or if the extraction of a larger amount of this metal is not objectionable. In admitting water from above, the outlet of the tank remains closed until the latter is full. The outlet is then opened, and leaching continued until the soluble salts are removed.

Leaching from Below.—Water is introduced from below the filter, either if the amount of silver extracted is to be diminished as much as possible, or if water is scarce, this method causing a saving of 25 to 40 per cent. wash-water, compared with the operation of leaching from above. The water is allowed to rise slowly through the charge until the tank is filled. Leaching from below is then stopped, the outlet of the tank opened, and leaching from above is commenced and continued until the soluble salts are removed. This method requires some further explanation. It has been shown in § 516 that a concentrated wash-water brine, saturated with silver chloride and salts of the base metals, if diluted, becomes turbid, and parts with some of these salts the more completely the further dilution is carried. If this dilution takes place in the ore-tank itself, the precipitated salts are arrested in the charge as soon as the current is reversed, and leaching from the top commences. Hence, the first wash-water will hold less silver in solution, but also remove a smaller amount of the base metals.

Drawing the Line.—When the first wash-water has done its work and has been replaced by lixiviation-solution, the appearance of which at the outlet of the tank must be carefully watched, the first part of the liquid running out is so weak in hyposulphite that it would dilute the stock-solution too much were it allowed to run to the regular precipitating-tanks. For this reason it is transferred to the weak solution precipitating-tanks until a certain concentration in hyposulphite has been reached (determined by the iodine-test) which, of course, depends upon the concentration of the stock-solution in hyposulphite. At the Marsac mill, for example, with a stock-solution of 2.0 to 2.2 per cent. concentration, the line is drawn when the solution leaving the tank has reached a concentration of 0.6 to 0.7 per cent. in hyposulphite. The number of inches of weak solution is noted and serves as a guide for drawing the line when the second wash-water is applied, so that the volume of the stock-solution remains constant.

Removal of the First Wash-Water by Draining Off.—In some mills the first wash-water is allowed to drain off as much as possible before commencing lixiviation. The outlet of the tank is then closed and the latter is filled with stock-solution. The tedious watching for the appearance of the first silver-bearing solution is thereby avoided, and in starting lixiviation the solution is at once taken to the precipitating-tanks. This practice is not permissible with ores that have a low rate of lixiviation because the charge settles more or less during draining and thus prevents easy filtration.

Quantity of First Wash-Water.—The quantity of first wash-water consumed is generally from 30 to 52 cubic feet per ton of ore, and more in some cases.

Leaching with Cold or Hot Water, and Silver Extracted by the First Wash-Water.—It has already been mentioned that roasted ores should be charged cold (see § 1301). This also implies that the first wash-water should be used cold. How materially the solubility of silver chloride is increased by hot wash-water brine, has been shown in § 517.

Table No. 1401. a. records results of leaching with water on a large scale at Cusihuiriachic. (Daggett.)

TABLE No. 1401. a.

Silver Dissolved from Cold and Hot Roasted Ores by the First Wash-Water Introduced from Above.

No. of Charges Experimented on.	Value of Ore in oz. Silver Per Ton.	Per cent. of Salts Soluble in Water.	Calculated Value in oz. Silver Per Ton.	Value after Leaching with Water in Mill. Oz. Silver Per Ton.	Ounces extracted by First Wash-Water.	Per cent. extracted by First Wash-Water.
HOT DRY ORE.						
5	34.6	14.8	40.6	28.3	12.3	30.3
33	34.2	14.1	38.8	27.6	11.2	28.8
COLD DRY ORE.						
19	37.9	15.0	44.6	39.4	5.2	11.6

Upon further examination it was found, however, that all this silver did not actually leave the ore-tank with the wash-water. More or less water, free from salts, exists under the filter of the tank, left there from the second wash-water of a previous charge. If the concentrated brine meets this water, silver chloride is precipitated under the filter, together with lead sulphate, *etc.*, and subsequently again more or less dissolved by hyposulphite solution. For Cusi ore, even when 25 to 30 per cent. of the silver in the ore had been dissolved and extracted from the ore by the first wash-water, only 3 to 9 per cent. actually left the tank charging the ore hot; while 2 to 6 per cent. left it when the ore was charged cold. Consequently, the difference between the results of hot and cold charging is not practically as great as would appear from Table No. 1401. a.

Table No. 1401. b. shows the apparent extraction of silver by the first wash-water from a number of charges at the Aspen mill. The ore was charged cold into the tanks filled with sufficient water to saturate the charge, and then leached from above.

TABLE No. 1401. b.

Silver Dissolved by First Wash-Water from Aspen Ore.

Number of Charges Treated.	Charge-sample Ounces Silver per Ton.	Soluble Salts. Per cent.	Silver dissolved by First Wash-Water. Per cent.
6	22.7	13.1	11.9
7	22.5	14.1	16.6
1	21.6	15.6	22.7

The actual extraction of silver by the first wash-water, *i. e.*, silver obtained in wash-water precipitates, averaged at the Aspen mill, on 30,000 tons of ore, 17.4 per cent. of the silver in roasted ore.

Table No. 1401. c. gives more information on the same subject.

TABLE No. 1401. c.

Silver extracted from Roasted Ores by the First Wash-Water, and the Value of the Precipitates.

Name of Mine.	Wash-Water Introduced from Above or Below.	Mode of Precipitating.	Per cent. of Ore-Value Obtained from Wash-Water.	Value of the Precipitate in Silver per ton.
Lake Valley	Above.	Acid and Scrap-Iron.	29.0	9000 oz.
San Bartolo	Below.	Sodium Sulphide.	3.3	1600 oz.
San Antonio	"	" "	6.4	1400 oz
San Miguel	"	" "	7.8	2400 oz.
Yedras	"	" "	2.0	13934 oz.

The result does not depend upon the richness of the ore in silver, but—outside of temperature—altogether upon the concentration and composition of the wash-water brine. The theory of this subject has been fully discussed in Chapter V., and does not require further elucidation.

Distribution of Silver in Wash-Water.—Table No. 1401. d. contains a statement as to the distribution of silver in the first wash-water, which was applied from above. It shows that at least 90 per cent. of the dissolved silver were contained in the concentrated brine leaving the tank during the first 15 minutes, after leaching with water had been commenced. (Daggett.)

TABLE No. 1401. d.

Distribution of the Silver Dissolved in the First Wash-Water from Cusi Ore.[*]

Value of Charges in oz. Silver Per Ton.	Per cent. of Salts Soluble in water.	Calculated Ore-Value in oz. Silver Per Ton.	Value after Leaching with Water in Mill.	Oz. Silver Extracted by First Wash-Water.	Silver Contained in 1000 c. c. at end of ¼ Hour.	Silver Contained in 1000 c.c. at end of One Hour.	Silver Contained in 1000 c.c. at end of Two Hours.	Silver Contained in 1000 c.c. at end of Three Hours.	Silver Contained in 1000 c.c. at end of Four Hours.	Silver Contained in 1000 c.c. at end of Five Hours.
29.2	14	33.9	25.2	8.7	.180	.0064	.0032	.0028	.0008	none
29.0	14	33.7	26.5	7.2	.108	.0034	.0020	.0009	trace	“
39.6	15	46.6	37.4	9.2	.150	.0116	trace	none	none	“

The Use of Chemicals in the First Wash-Water.—Numerous experiments have been made at the Aspen mill, Colorado, to improve the extraction of roasted ores by dissolving chemicals in the first wash-water, namely, soda-ash, sodium bi-carbonate, blue-stone, nitre and sulphuric acid, and bleaching-powder and sulphuric acid. While this mode of treatment has, in most cases, been efficient in reducing the "going back" of the chlorination of the washed ore, it did not materially influence the final extraction of the silver.[†] In one of these experiments, however, a favorable financial result was obtained, namely, by dissolving soda-ash in the first wash-water. The results of this experiment are given in Table No. 1401. e. The mode of treatment was as follows: The ore-tanks, of 17 ft. diameter and 9 ft. depth, were filled with 70 inches of water in which the soda-ash was dissolved. Into this water a charge of 60 tons of cold ore was dumped, and then followed by 70 inches more water. The charge was then treated in the usual way, *i. e.*, by 1st ordinary; 1st extra; 2d ordinary; 2d extra; 3d ordinary solution. The quantities of these solutions varied more or less to reach a satisfactory final extraction. In the table below 1 inch solution = 18.9 cub. ft.; or 1 inch solution = 0.315 cub. ft. per ton of ore.

---

[*] The silver in the wash-water is expressed in grammes per 1000 c. c. The depth of the charges in ore-tanks was 24 inches. With deeper charges the results would have been different as to time of leaching.

[†] Blue-stone dissolved in the first wash-water is beneficial in counteracting the deleterious influence of caustic lime in roasted ores.

The chemistry of the process is as follows: Copper in roasted ore is principally present as cuprous chloride. The first wash-water extracts most of this copper, cuprous chloride being soluble in brine. If sodium carbonate is present in the wash-water, cuprous carbonate is precipitated, retained in the charge, and readily dissolved when the first ordinary solution is applied, thus forming a weak extra-solution. From cupric salts the soda-ash precipitates cupric carbonate. To what extent this salt is dissolved by the first ordinary solution, I do not know. It will be remembered that if to a hyposulphite solution, containing sodium carbonate, a cupric salt is added, cupric carbonate is precipitated, showing that the latter salt is not easily soluble in hyposulphite solutions.

The mill-run was made on 24 charges of 60 tons each, the ore being taken from the same ore-bed. Its composition was as follows:

Silver, 23.8 ounces per ton; SiO_2, 16.12; $BaSO_4$, 23.77; CaO, 11.23; MgO, 4.99; Pb, 2.8; Fe, 9.89; Zn, 3.4; Cu, 0.48; S, 9.07 per cent.

The roasted ore contained on an average 12.6 per cent. soluble salts.

Cost of blue-stone is figured at 5 cents, and that of soda-ash at 3.5 cents per pound.

The following facts become apparent from Table No. 1401. e.:

1st. In the charges without soda-ash in the first wash-water, the chlorination of the washed ore went back 21.2 per cent. The average "going back" of the charges using soda-ash was only 14.9 per cent.

2d. The apparent extraction by 1st ordinary in mill averaged 51 per cent. using soda-ash in wash-water, against 17.8 per cent. using water only.

3d. Although the final silver-extraction in the mill was practically the same, a considerable saving in cost of chemicals was effected by using wash-water with 0.3 and 0.1 per cent. of soda-ash, this saving being 20.1 and 22.3 cents per ton of ore respectively.

TABLE No. 1401. e.

Results of Mill-Run treating 24 Charges of 60 Tons each without and with Soda-Ash in First Wash-Water.

Number of Charges.	Per cent. of Soda-Ash in First W.-W.	Charge-Sample. Per cent. Silver extracted in Assay-Office.		Washed Ore. Per. cent. Silver extracted in Assay-Office.		Per cent. Apparent Extraction in Mill.			Inches Solution per Charge.	Cubic Feet per Ton.	Per Ton of Ore.				
											Soda-Ash.		Blue-Stone.		Total. Cents.
		Ord. Sol.	Extra-Sol.	Ord. Sol.	Extra-Sol.	Wash-Water.	First Ord.	Final Extraction.			Pounds.	Cents.	Pounds.	Cents.	
6	none	76.7	91.2	55.5	88.0	11.9	17.8	87.0	459	144.5	none	....	16.3	81.5	81.5
3	1.0	77.8	91.8	62.0	89.7	12.5	49.8	88.2	361	113.7	13.5	47.2	10.8	54.0	101.2
7	0.5	79.8	90.7	59.8	89.5	8.9	50.7	87.1	415	130.7	6.75	23.6	11.8	59.0	82.6
5	0.3	87.1	87.8	71.7	87.7	9.9	51.6	87.1	470	129.1	4.0	14.0	9.4	47.4	61.4
3	0.1	83.8	89.3	75.5	89.3	7.1	52.0	87.5	481	151.5	1.35	4.7	10.9	54.5	59.2

Wash-Water Precipitates below the Filter of Ore-Tanks.—With some ores a perceptible quantity of precipitates from wash-water is formed below the filters of the ore-tanks. These precipitates are partly carried forward by the wash-water, and then settle at the bottom of the wash-water launder. An average sample taken at the Marsac mill, contained 47.8 per cent. lead, and 1649 ounces silver and 0.45 of an ounce gold, per ton. The hyposulphite solution, following the first wash-water, does not completely dissolve that portion of the wash-water precipitates which remains below the filter, but carries most of the undissolved portion into the solution-launder. Although these precipitates, after settling at the bottom of the solution-launder, are constantly exposed to the hyposulphite solution, they retain more or less silver. A sample, taken at the Marsac mill, contained, according to W. G. Lamb, 70 ounces silver, per ton; and another sample, which, most likely, had been exposed to the solution for a shorter time, assayed 524 ounces silver, per ton.

A sample from the wash-water launder, with 1597 ounces silver and 42.2 per cent. lead, was treated in the assay-office with ordinary solution in the usual way for making chlorination-tests. Only 25.6 per cent. of the silver were extracted.

TREATMENT OF THE SILVER-BEARING WASH-WATER.

The extraction of silver can be effected by two methods.

§ 1402. *By Precipitation with Sodium or Calcium Sulphide.*

This method especially recommends itself for wash-water obtained from ores containing caustic lime. Such wash-water carries a comparatively small amount of base metals, and will yield sulphides of high grade, as shown by the Yedras product, Table No. 1401. c.

At the Aspen mill, laboratory experiments and mill-results have demonstrated that silver can be completely precipitated from wash-water by stirring it with sulphides obtained from a previous charge by precipitation with sodium sulphide. This method not only saves sodium sulphide, but also produces sulphides of higher grade in silver. The process is based upon the following reaction: Base metal sulphides precipitate silver sulphide from a silver chloride solution.

§ 1403. *By Precipitation with Copper or Iron.*

Most applicable is this method for wash-water from acid ores containing a large amount of base metals. In the presence of a large percentage of copper relatively to silver, it is most profitable to use copper as precipitant. From the decanted solution copper is then precipitated by scrap-iron.

If comparatively little copper is present in the solution, iron is used as precipitant, the cement-copper resulting at the same time greatly facilitating precipitation of the silver. In each case sufficient sulphuric acid should be added to give to the solution a decided acid reaction. Heating of the solution also hastens the reaction. The process is best conducted in the apparatus and manner described in § 1102.

In conclusion I advise not to run to waste solutions decanted from wash-water and weak solution precipitating-tanks, but to collect them in large tanks and allow them to undergo there a second precipitation and settling. This practice has been found very profitable at the Aspen mill, Colorado. The tanks were partly filled with scrap-iron; very weak sodium sulphide solution was also used.

THE LIXIVIATION.

The mode of treatment with hyposulphite solutions varies according to the character of the ore, whether it is raw or roasted, and in the latter case, whether it is acid, arsenical alkaline, or alkaline.

In ordinary lixiviation there is very little variety possible in conducting the process. With the Russell process, on the contrary, so many modifications become possible that their number is unlimited. In the following, some general instructions are given about different classes of ores, based upon practical experience.

§ 1404. *Lixiviation of Raw Ores.*

A standard extra-solution is generally used first, with [or without] circulation, followed by ordinary solution. In rare instances the standard extra-solution is preceded by ordinary solution. In this, and in other cases, it is not always possible to give "the reason why." Try different methods and select the best.

How standard extra-solution is prepared for raw ores, has already been described in § 1203.

The difficulty in lixiviating raw ores is often a purely mechanical one, *i. e.*, the solutions will not filter through the charge. As an example I cite the lime-ores of Aspen, Colorado. Numerous laboratory experiments showed that as much as 90 per cent. of the silver could be extracted from some of these ores by extra-solution. Ore from the Red Spruce mine, assaying 22 ounces silver, per ton, was pulverized so as to pass screens of various sizes, and then lixiviated with extra-solution in the laboratory with the following results :

Ore Crushed to pass :	*Per cent. Silver Extracted with Extra-Solution.*
No. 80 Screen.	86.8
" 60 "	86.4
" 40 "	84.8
" 30 "	85.2

It will be noted that there is no material difference in extraction between ore crushed through No. 80 and No. 30 screen.

Eight and a half tons of this ore were crushed in the Aspen mill through a No. 30 screen, and charged into an ore-tank, filling the latter to a depth of 14 inches. The tank had been previously filled with 36 inches of extra-solution containing 1 per cent. of blue-stone and 2.25 per cent. hyposulphite. It was found that the solution would not filter. (The ore, after roasting, has a rate of lixiviation of 13 inches). After stirring the charge for four hours it was allowed to settle clear and the solution decanted. The ore was then washed twice with water by decantation. The tailings assayed 7 ounces, showing an extraction of 69 per cent. of the silver. It is evident from this that an extraction equal to the laboratory-test would have been obtained if the character of the ore had not prevented filtration. It might have been possible in this case to produce a small rate of lixiviation by a Koerting ejector.

§ 1405. *Lixiviation of Acid Roasted Ores.*

Ordinary solution always precedes standard extra-solution—generally circulated—which is followed again by ordinary. Here I have to add some remarks about the volume of standard extra-solution. It has been stated, in § 1203, that this volume is taken just enough to saturate the charge. If extra-solution follows ordinary solution, and is circulated, its volume is taken about 10 per cent. short of that required for saturation. It can readily be seen that it would be to no purpose, in displacing ordinary solution by standard extra-solution, to run a part of the latter into the precipitating-tanks before it has been circulated and done its work, which might occur if the volume were taken too large.

Effect of Circulation.—That circulation of standard extra-solution may be very essential for acid roasted ores in producing maximum results, becomes evident from Table No. 1405. The same experience has been made with this class of ores in other mills.

TABLE No. 1405.

Effect of Circulating the Extra-Solution as Compared with Not Circulating, (Acid Roasted Ore) at Cusi Mill.

"Circulated" or "Not Circulated."	Number of Charges Treated.	Value of Ore in Silver Per Ton. Ounces.	Per cent. Extraction by Extra-Solution in Assay Office.	Per cent. Extraction in Mill.
The extra-solution stood in the ore for twelve hours.	46	34.8	89.0	78.2
The extra-solution was circulated for four hours.	96	34.8	87.6	85.1

§ 1406. *Lixiviation of Arsenical Alkaline Roasted Ores.*

The standard extra-solution is not circulated, but allowed to stand in the charge from 10 to 12 hours. It may then be circulated. This is a matter of experience, and I am not able to offer an explanation. That the difference in results between circulating and not circulating the standard extra-solution is considerable, is shown in Table No. 1406. It is interesting to compare this table with the preceding one.

TABLE No. 1406.

Effect of Allowing the Extra-Solution to Stand Twelve Hours in the Ore, as Compared with Circulating the Same (Alkaline Arsenical Roasted Ore), at Yedras Mill.

"Circulated" or Allowed to Stand Twelve Hours."	Value of Ore in Silver Per Ton. Ounces.	Per cent. Extraction by Extra-Solution in Assay Office.	Per cent. Extraction in Mill.
The extra-solution was circulated for four hours.	57.4	83.3	77.7
The extra-solution stood in the ore for twelve hours.	63.5	83.1	82.6

§ 1407. *Lixiviation of Alkaline Roasted Ores.*

If roasted ores, containing an appreciable percentage of caustic lime, are treated, the extra-solution should, in most cases, follow the first wash-water. That a good effect is produced by dissolving blue-stone in the wash-water, into which the charge is dumped, has already been stated previously.

It has also been found beneficial to impregnate the ore with a small quantity of a strong copper sulphate solution before applying the extra-solution. The *modus operandi* is as follows: About one or two pounds of copper sulphate per ton of ore are dissolved in such a volume of water that it will fill the ore-tank to a depth of two or three inches.

As soon as the first wash-water has sunk to the surface of the ore, leaching with water is suspended, and the copper-solution is put on. The latter is made to sink a little below the surface of the ore before the extra-solution is run on the charge, and lixiviation is started. When the extra-solution has displaced the wash-water, the discharged liquor is turned into the precipitating-tank, and lixiviation with extra-solution is continued. The extra-solutions used for alkaline ores are neither of standard composition, nor are they circulated.* They are operated like ordinary solution, and contain, as already stated in § 1203, an excess of sodium hyposulphite above standard composition, besides being of low concentration in copper. These extra-solutions are sometimes put on in several charges, commencing with one of $\frac{33}{100}$ per cent. copper sulphate, followed by others with less copper sulphate, the last one containing only $\frac{1}{10}$ per cent. of this salt. After this, the ordinary solution follows.

It is hardly necessary to say in conclusion that the most suitable method of treatment for a given ore cannot be stated *a priori*, but must be established by careful experimenting.

Thus, for example, it may be advantageous, as has been done at Aspen and elsewhere, to treat a charge first with ordinary solution and then several times in succession with extra-solution followed each time by ordinary solution. The general statements found above will, however, serve as a useful guide in starting the experiments.

§ 1408. *Distribution of Silver and Lead in the Solution during different Periods of Lixiviation.*

Russell made experiments, at the Ontario mill, to ascertain the distribution of silver and lead in the solution during the progress of lixiviation. He found that the first one-sixth of the solution, running out of the ore-tank contained 44.7 per cent. of the silver finally extracted, while the following portions decreased rapidly in their contents in silver. The last one-twelfth extracted only about 1 per cent. of the silver. The contents of the solution in lead, on the contrary, were nearly the same during the whole progress of the operation.

In treating a charge of 60 tons of roasted ore at the Aspen mill, the following quantities of solutions per ton of ore, applied in succession to the charge, extracted silver as follows :

---

* In treating Aspen ore, for example, it was found that by circulating extra-solution silver was left in the charge which had been previously extracted.

			Per cent. Silver extracted.
36.2	cub. ft.	wash-water	22.7
47.2	"	1st ordinary	32.0
25.2	"	1st extra	30.1
31.5	"	2d ordinary	
12.6	"	2d extra	1.7
15.7	"	3d ordinary	
12.6	"	3d extra	4.1
12.6	"	4th ordinary	
		Total extraction	90.6
		Remaining in tailings	9.4

During the progress of lixiviation of this charge, samples of the solution leaving the tank were taken for every 20 inches of ordinary and extra-solution passing through the ore. In these samples silver was determined per 100 c.c. of solution. Samples were also precipitated with Na_2S, and the sulphides analyzed for silver, copper, lead and sulphur. The solution was highest in silver after 40 inches of ordinary solution had passed, namely, 37.2 mgm. per 100 c.c.; the sulphides were also highest in silver, namely, 13,104 ounces per ton. The solution then became rapidly poorer in silver, declining to 5.5 mgm. per 100 c.c., the silver in precipitated sulphides going down, at the same time, to 3290 ounces per ton. The 1st extra brought the solution up again to 13.6 mgm. silver. After this the silver in solution declined rapidly, and was only 0.90 mgm. at the end of 2d ordinary. The 2d extra brought it up again to 1.40 mgm.; and then came a steady decline in value. The sulphides precipitated from the solution became of very low grade, and assayed finally only 300 to 132 ounces silver, per ton. The average silver-contents of the solutions, first wash-water included, passing through the charge, were 9.365 mgm. per 100 c.c.

W. S. Morse has observed that whenever extra-solution is turned on the charge, the amount of lead dissolved is greatly reduced as compared with the effect of ordinary solution, and the reduction is most marked the more the extra-solution approaches standard composition.

This is easily explained: the extra-solution acts like a weak ordinary solution in its solvent energy on lead sulphate, no reaction taking place between this salt and cuprous hyposulphite.

§ 1409. *Method of Determining whether the Extraction of the Silver is completed.*

If lead-bearing ores are treated, the facts recorded in § 1408 are of importance for two reasons. It is evident that lixiviation should be suspended as soon as the silver is extracted, a continuation of the process furnishing lead only. The reaction with sodium sulphide cannot be relied upon to determine whether the silver has been extracted from a charge, the lixiviation-solution giving a copious precipitate of lead sulphide. Even in the absence of lead, this reaction is not a safe guide, because the stock-solution, unless precipitation of the metals has been complete,—which should not be the case—always retains some silver and copper.

The only reliable way to ascertain whether the extraction of the silver is completed, is to assay the tailings, which should show a normal value, established by experience, nearly corresponding with the tailings obtained by assay-office lixiviation-tests. Very soon, after starting a mill, the volume of solutions necessary for best results becomes known, and investigation as to value of tailings becomes thereby simplified. That a change in character of the ore should be carefully watched, is hardly necessary to mention.

CHAPTER XV.

Trough Lixiviation.*

O. Hofmann claims priority in having discovered that all the silver chloride in roasted ore is dissolved by hyposulphite solution, in $\frac{3}{4}$ to $1\frac{1}{2}$ minutes, if the ore is introduced into a running stream of the solvent. It is rather quantity than concentration of the solution that produces this effect. The proportions of liquid and ore vary, and have to be determined by experiment in each case. Hofmann found, for instance, that this proportion for Cusihuiriachic ore was >12 to 1, and <18 to 1. To this method he has given the name of trough-lixiviation, because the solution is effected in triangular, wooden troughs. The latter have a minimum length of 150 feet, with a fall of at least $\frac{3}{4}$ of an inch to the foot. Two separate plants are required, one for base-metal leaching, and one for silver-lixiviation. The *modus operandi* is as follows: The ore is continuously charged into an agitator which breaks up lumps, and discharges into the base-metal leaching-trough. The latter ends above a system of settling-tanks, connected with each other, and so arranged that any one tank can receive the charge from the trough, and can be disconnected without interrupting the operation of the whole system. Since the same arrangement is used in wet-crushing of silver-ores, a detailed description is not necessary. From the tank which is, at the time, the last one, the clear wash-water runs to waste. These tanks are provided with filter-bottoms, and with central discharges for sluicing the ore. As soon as a tank is filled, it is disconnected, allowing the pulp to flow into the next one. Fresh water is introduced upon the ore in the full tank to displace the base wash-water, and, finally, hyposulphite solution to displace the water. The tank is now ready for sluicing with hyposulphite solution. The silver lixiviation-trough leads to a system of settling-tanks of precisely the same arrangement and construction as the one described above, and the clear solution, charged with silver, runs into the precipitating-tanks. Whenever an ore-tank is full, the hyposulphite solution is displaced by water, and the tank becomes ready for sluicing the tailings. Hofmann claims a great many advantages for this system, considering it superior in every respect to the ordinary practice. In doing so, he loses sight of the mechanical difficulties which this method presents. It can not be assumed that the number of tanks will be lessened, as compared with the customary system, if the solutions are to be discharged perfectly clear. Such a volume of wash-water is obtained in the base-metal leaching that it becomes impracticable to regain from it any silver it may hold. Hofmann states that no loss of silver will be incurred from this source. He bases his statement upon the fact that, according to Fresenius, silver dissolved in concentrated solutions of the chlorides of sodium, calcium, etc., is completely separated on sufficient dilution with cold water. Silver will, no doubt, go in solution when the ore is charged into the agitator. Even granting that this silver is again precipitated on sufficient dilution, it is questionable whether the fine particles of silver chloride will settle in the tanks, or float off with the wash-water.

---

* Eng. and Mg. Journal, Sept. 10th and Nov. 26th, 1887. Trans. A. I. M. E., Vol. XVI, 662.

According to Chaney,* assayer at the North Mexican mill, while Hofmann experimented with trough-lixiviation, the sulphides from two mill-runs of 450 and 300 tons, assayed 2953 ounces, and 1822 ounces silver, per ton, respectively, whereas sulphides, produced by ordinary lixiviation from ore of the same character, assayed from 9506 ounces to 15877 ounces silver, per ton. Chaney states that these low-grade sulphides were produced by precipitating the lixiviation-solution in a turbid condition, it being impracticable to allow sufficient time for a complete settling of the finest slimes before precipitation.

The greatest mechanical difficulty, however, will be met in the handling of the tanks. It can readily be seen that as soon as a system of tanks is in complete operation, each one will be filled with so many strata of ore of different grades of fineness as there are tanks in the system, less one. The very finest slimes rest directly upon the filter, succeeded by strata of coarser and coarser material. The last stratum, which is formed when the ore enters the tank directly from the trough, is, of course, the largest one in thickness, all other strata taken together occupying much less depth. Besides this, the deposition of the ore on the filter is irregular. Where the trough discharges the ore into the tank, the coarsest particles will drop to the bottom in one spot, thus making filtering in this particular place easier than elsewhere. Hence, in displacing, finally, the lixiviation-solution by wash-water, the latter will find its way through that portion of the charge offering the least resistance, without permeating the finer strata. Thus, the pressing out of the lixiviation-solution may be apparently finished, while, in fact, a large portion of the solution is left in the ore, and is lost. Since filtering cannot be dispensed with in trough-lixiviation, this operation will not only become irregular, but also very slow. Before Hofmann can show some decided success in practice, trough-lixiviation must be considered as one of the many processes that figure on paper only.

---

*Eng. and Mg. Journal. Dec. 3d, 1887.

The fact that in lixiviation, not the solution of the silver, but the filtering takes time, was known long before Hofmann invented trough-lixiviation. E. H. Russell, while experimenting with the raw lixiviation of Bullionville tailings, wrote to me, on Jan. 3d, 1885, as follows: "In treating Bullionville tailings, we charge them in an agitator with muller revolving 28 times per minute. This agitator, which at the same time receives the hyposulphite solution, breaks up lumps, and discharges the tailings into a trough leading to the ore-tanks. *All the silver* (that can be extracted) *is practically dissolved by the time the tailings reach the tanks*, but the difficulty is to get the solution through the filter, and, especially, to displace the solution by wash-water, etc. . . . ."

The same observation was made at the Flagler Reduction-Works, Silver City, New Mexico. Others, however, had observed these facts before Russell. Hence, Hofmann cannot claim priority in having discovered the principle upon which trough-lixiviation is based. In the unfavorable view I take of trough-lixiviation I am supported, so far as able to ascertain, by all who have had practical experience in lixiviation.

Ellsworth Daggett, in his paper on the Russell process, adds valuable criticism. He says:

"In the *Engineering and Mining Journal* of Nov. 26, 1887, the statement is made that for Cusi ore the minimum amount of stock-solution required for trough-lixiviation was 12 parts of solution to 1 of ore, which is 384 cubic feet per ton, although the strength of the solution in sodium hyposulphite was 1.6 per cent. The average amount of stock-solution used per ton of this ore in the usual method of lixiviation, for a period of 9 months, and with a solution of 0.75 per cent. sodium hyposulphite, was only 80 cubic feet. Hence, trough-lixiviation, using a solution twice as strong, and requiring 4.8 times the volume, necessitated the use of 9.6 times as much sodium hyposulphite per ton of ore. One of the evil results of this would be the production of a lower grade of sulphides, and an excessive use of precipitant, as the amount of lead dissolved is directly proportional to the amount of sodium hyposulphite used. By referring to the account of some Yedras experiments in the same journal of Jan. 21, it will be seen that the injurious effect of caustic lime was the same while using trough-lixiviation as while lixiviating by the usual method."

That the increased volume of stock-solution requires more labor in precipitation, and causes greater loss in sodium hyposulphite, is hardly necessary to point out.

Since writing the above, in 1888, trough-lixiviation has gone entirely out of existence, unless I am greatly mistaken, so that my criticism, to which Hofmann greatly objected, must be founded on facts, and not on fiction or prejudice.

CHAPTER XVI.

THE PRECIPITATION OF THE METALS FROM A LIXIVIATION-SOLUTION IN THE MILL.

§ 1601. *Constitution of a Lixiviation-Solution before Precipitation.*

The lixiviation-solution may hold, outside of silver and gold, the following elements that play an important part in precipitation, namely: Copper, lead, antimony, arsenic, and calcium. Neither iron, zinc, nor manganese can be present in perceptible quantities if the first wash-water has been properly applied.

Copper is derived from extra-solution, and from cuprous chloride in roasted ores; lead is introduced through lead sulphate in raw and roasted ores; antimony and arsenic are derived from antimonates and arsenates in roasted ores; calcium comes from calcium sulphate and caustic lime in roasted ores, and from gypsum in raw ores, it being difficult to remove an excess of calcium sulphate and hydrate entirely by the first wash-water from roasted ores. [It is here assumed that sodium sulphide is used as precipitant for silver.] Although the quantity of the base metals in the lixiviation-solution rarely exceeds a few pounds per ton of ore, the value of the precipitated sulphides in precious metals may nevertheless be thereby diminished to a considerable extent.

§ 1602. *The Precipitation of Lead [and Calcium] with Sodium Carbonate.*

Manipulation.—The precipitation of lead is easily effected. While adding the soda-solution, the contents of the tank are stirred by compressed air. The end of the reaction is best determined by taking a sample of the clear solution from the precipitating-tank in a test-tube, and adding a few drops of sodium phosphate, which gives a more voluminous lead-precipitate, and a more delicate reaction than sodium carbonate. After complete precipitation of the lead, the solution is stirred for a few minutes longer. The lead carbonate, being more compact than a precipitate of sulphides, settles quickly, and in less than half an hour the clear solution is decanted, and transferred to the silver precipitating-tanks. The lead carbonate is removed from the tank about once a week, and made into cakes by a filter-press.

Quantity of Lead dissolved in Lixiviation.—In § 224, it is stated that the solubility of lead sulphate is materially decreased if a lixiviation-solution is of low concentration in sodium hyposulphite. A secondary influence has the temperature of the solution, a warm one being a better solvent than a cold one. Since the stock-solution rarely exceeds 1½ per cent. concentration in sodium hyposulphite, it follows that but a limited amount of lead is actually dissolved, even if the ore contains several per cent. of lead sulphate. In practice, the amount of lead extracted can be kept below 5 pounds per ton of ore, if proper precautions are taken.

§ 1603. *The Precipitation of Silver, Gold, Copper [and Lead.]*

Manipulation.—The manipulation is the same, whether sodium or calcium sulphide is used.

In adding sodium sulphide to the solution in the precipitating-tank, the contents of the latter are well stirred by compressed air, and this operation is continued for some time after precipitation is completed. The more violent the stirring, the quicker the settling of the sulphides, and the clearer the solution. The sodium or calcium sulphide solution is either measured in a bucket, or, in case a solution-pipe runs to the precipitating-tank, its flow, with cock fully open, is timed by counting the strokes of a pendulum. (Arents.)

In this process silver is precipitated first and the base metals subsequently; but the reaction is not sufficiently close to take advantage of it in a practical way. Although very high-grade sulphides would result from partial precipitation, those obtained by further precipitation would not be free from silver, and nothing would be gained by the operation. As a rule, close precipitation is most advantageous because it fully restores the solvent energy of the stock-solution. The beneficial effect of occasional over-precipitation has already been discussed in § 605.

In some mills, for example at Yedras, ordinary and extra-solution are precipitated separately.

Testing for the End of the Reaction.—The end of the reaction is recognized if a sample of the clear solution in a test-tube gives only a slight precipitate with sodium sulphide. Of course, sodium sulphide must not be added in excess to the solution in the precipitating-tank; a sample of the clear solution should not give a reaction with argentic hyposulphite.

Over-Precipitation.—If over-precipitation has taken place, more lixiviation-solution must be run into the tank; for this reason, the latter should not be completely filled before commencing precipitation. The addition of an excess of sodium sulphide is sometimes indicated by the odor of sulphuretted hydrogen.

It may occur that, by inadvertence, an over-precipitated solution is transferred to the storage-tanks; then, silver sulphide will be precipitated in the ore-charge in running such a solution to one of the ore-tanks. By immediate treatment with extra-solution, this silver is extracted—one of the advantages of the Russell process. Formerly, with ordinary lixiviation, such a charge had to be returned to the roasting-furnaces.

Decanting the Solution.—After the sulphides have settled, the clear solution is decanted by the swinging pipe into the solution-sumps.

The settling of the sulphides requires from 1 to 4 hours.

Value of Sulphides.—The value of sulphides in silver may vary between 3,000 and 12,000 ounces per ton, much higher values being of rare occurrence.

§ 1604. *Mechanical Handling of the Sulphides.*

The sulphides should be removed from the precipitating-tanks at least every third day. After decanting the clear solution as low as possible, the precipitatate is discharged into the storage-tank for sulphides. Here, settling takes place again, and clear solution is decanted so as to save running it through the filter-press. The sulphides may be reduced in bulk by introducing steam into the storage-tank. If this is done, the mass should be allowed to cool before running it into the filter-press, because a warm solution destroys the filter-cloth much quicker than a cold one. The sulphides are now ready for the press.

Disadvantages of Leaving the Sulphides too long in the Precipitating-Tanks.—O. Hofmann condemns the practice of removing the sulphides only once a week. Freshly precipitated sulphides, he says, form large flakes which settle easily. After several days, they assume a dry and sandy condition, and, if again stirred up, divide into very fine particles which are kept suspended in the solution for a long time. The result of such a practice is that the decanted solution retains silver sulphide in suspension when returned to the storage-tanks, and silver sulphide covers the surface of the ore when the solution is used for lixiviation.

Manipulation of the Pressure-Tank and the Filter-Press.—It is not necessary here to describe the manipulations of the filter-press, instructions being furnished by the manufacturer.

Moisture in Sulphide-Cakes.—Sulphides obtained at 80 pounds pressure appear quite dry, but contain from 38 to 42 per cent. moisture. At higher pressure the moisture is considerably reduced.

Drying of Sulphides.—It is of importance to remove from the cakes of sulphides, as they come out of the filter-press, every trace of moisture in the drying-chamber. Only sulphides entirely free from moisture can be correctly sampled after they have been put through the sample-grinder.

CHAPTER XVII.

The Refining of Sulphides.

The refining of sulphides at the mill is one of the great disadvantages of lixiviation compared with amalgamation. The methods of the smelters cannot be used to advantage, because of the large percentage of silver accumulating in by-products, which can be treated cheaply by smelting, but are not suitable for mill-work.

§ 1701. *The Old Practice.*

The old practice of treating the sulphides consisted in roasting them in a reverberatory-furnace, and either melting the roasted material in black-lead crucibles with scrap-iron and borax, or in cupelling it with lead. The roasting was not carried to a dead-roast, because by converting all the copper sulphide into oxide, fusion in a crucible would become difficult; hence, sufficient sulphur was left to form a matte. Scrap-iron was added to reduce, as much as possible, the contents of this matte in silver. If lead was present, it was, in part, precipitated by the iron, and assisted materially in desilverizing the matte. In spite of this, a large percentage of the silver remained in the by-products. As an example I cite the experience at the Cusihuiriachic mill, Mexico.

Crucible-Melting at Cusihuiriachic.—Of the total silver in the roasted sulphides, according to Frank Johnson, crucible-melting returned 81.8 per cent. in bars, and 18.2 per cent. in by-products.

Cupellation at Cusihuiriachic.—The cupelling-process has the advantage of turning out refined bullion, but as to silver remaining in by-products, it is not much superior to crucible-melting. At the Cusihuiriachic mill, 83.1 per cent. of the silver was returned in fine bars, 14.2 per cent. remained in by-products, while over 2 per cent. was lost in cupellation. The cost of crucible-melting was $1\frac{94}{100}$ cents, and of cupellation $2\frac{16}{100}$ cents per ounce of fine silver. To this must be added the cost of shipping the by-products to smelters, the smelting-charges, and discount on the silver in the by-products. The more copper the sulphides contain in proportion to the silver, the greater will be the loss in by-products and the expense of refining.

Loss of Silver in Roasting.—The loss of silver in roasting the sulphides in a reverberatory-furnace may be very great. This loss is, in part, a mechanical one, but, principally, one by volatilization. Charging the furnace with moist sulphides increases the loss. Also the material of which the roasting-furnace is constructed absorbs considerable silver, which, however, may not be reckoned a total loss. An examination of this subject at the Cusihuiriachic mill, by Frank Johnson, showed that as much as 6 per cent. of the silver was lost in roasting, charging the sulphides dry; and as much as 12 per cent., charging them moist.

These methods are now out of use, most lixiviation-mills finding it more profitable to sell their dry and unroasted precipitates to smelters.

Refining Sulphides by Humid Processes.—Recently, several humid processes have been proposed for refining sulphides at the mill. Of these only two have been introduced, namely, the Stetefeldt process and the sulphuric acid process, and these alone will be considered here.

§ 1702. *The Stetefeldt Process for Refining Sulphides.**

The process described below (and which is not patented) was introduced by the author, in 1891, at the Marsac mill, Park City, Utah.

The Marsac lixiviation-mill produces three classes of precipitates:

1. Base sulphides, obtained in precipitating the silver-bearing wash-water by sodium sulphide. They contain, after drying: 2200 to 5200 ounces of silver and 1.6 to 3.9 ounces of gold, per ton, with 2.6 to 5.2 per cent. of copper and 38 to 45 per cent. of lead.

2. Carbonates, obtained in precipitating the hyposulphite solution by Solvay soda. They contain, after drying: 600 to 1400 ounces of silver and 0.6 to 1.9 ounces of gold, per ton, with 23 to 41 per cent. of lead.

3. So-called "regular" sulphides, obtained in precipitating hyposulphite solution (after precipitation by Solvay soda) with sodium

---

* Trans. Am. Inst., Vol. XXI, 286.

sulphide. They contain, after washing and drying: 11,200 to 12,000 ounces of silver and 11.1 to 12.1 ounces of gold, per ton, with 23 to 28 per cent. of copper and from a trace to 1 per cent. of lead. About 85 to 87 per cent. of the total silver produced by the mill is concentrated in the regular sulphides.

The base sulphides and carbonates are not treated in a washing-filter-press, and consequently hold a considerable quantity of soluble salts, from 10 to 13 per cent.

A sample from a lot of "regular" sulphides, not washed in the filter-press, was analyzed by Stalmann with the following results:

	Per cent.
Copper	21.60
Sulphur	24.83
Iron	0.75
Alumina	0.25
Arsenic	Trace
Antimony	0.18
Silica	0.25
Lead	0.50
Silver	34.78
Gold	0.03
Soluble in water	12.76

The alumina and silica are derived from tailings-slimes suspended in the silver solution; the soluble salts come from the hyposulphite stock-solution.

The base sulphides and the carbonates are shipped and sold to smelters. The regular sulphides only are refined at the mill, because sulphides, with a large percentage of lead, would not be suitable for treatment.

The process consists in the following operations:

1. Matting the sulphides in an iron pot.

2. Roasting the pulverized matte in a muffle-furnace.

3. Dissolving the roasted matte in dilute sulphuric acid.

4. Crystallizing from the solution blue-stone, which is used in the mill for preparing Russell's extra-solution.

5. Washing the silver-residue, pressing it into cakes, and melting the dry cakes to bars.

THE PLANT.

Matting-Furnace.—The cast-iron pot is 3 feet by 2 feet on the top, somewhat contracted at the bottom, and 11 inches deep. The bottom is 2 inches thick, and the sides taper to a thickness of 1 inch on the top. It is set in a fire-place, and covered by a hood of sheet-iron, provided with a working-door. A stove-pipe connects the hood with the Roessler converter through which the escaping gases are drawn. These pots should be of fine-grained iron, and entirely free from flaws. Those we had made at the Risdon Works, San Francisco, were by no means free from blow-holes; still, we succeeded in matting 15,000 pounds of sulphides in one of the pots before it commenced to leak.

The Brueckner Pulverizer.—For pulverizing the matte, I imported a small Brueckner ball-mill, with drum 800 mm. in diameter, and 500 mm. wide, made by the Sachsenberg Brothers, Rosslau, Germany. It is exceedingly well constructed and suited for pulverizing silver-copper matte. Large machines are used for this purpose at the Ziervogel extraction-works, Mansfeld, Germany. The pulverizer makes 28 revolutions per minute, requires about 1½ H. P. to run it, and pulverized about 150 pounds of matte per hour through a No. 40 brass wire-screen. The wear and tear is hardly perceptible. Forged wrought-iron balls are used. These were on hand at the Ontario mill from a discarded Brueckner pulverizer, erected many years ago. That machine was put up for pulverizing salt, which it did very well, but it had been so poorly constructed in one of the San Francisco foundries that it broke down at least once every 24 hours. Brueckner's pulverizer would have been as great a success in the United States as it is in Germany if well-constructed machines of the Sachsenberg type had been put on the market.

The Muffle-Furnace.—The roasting of the pulverized matte is done in a muffle-furnace, the muffle being heated from the top only. The hearth of the muffle is oval in shape, 7 feet long, and 4 feet 6 inches wide, the bottom being formed by a cast-iron plate 1¼ inches thick, with rim 1½ inches high. There are two working-doors on opposite sides of the muffle. An air-slit, next to the fire-place, supplies with oxygen that part of the muffle which lies between the fire-place and the working-doors. The fire-place, 3 feet 6 inches by 18 inches, is arranged to burn coal.

The end of the muffle is connected by iron gas-pipes, four inches in diameter, with the Roessler converter. To avoid overheating of the Koerting blower, made of hard lead, the pipes are cooled by a spray of water. In the center of the hearth is a discharge-opening, 6 inches square, closed by a well-fitting cast-iron plate. A car stands below this opening. The charging is done through the working-doors.

The Roessler Converter.—This consists of a lead-lined tank, 4 feet in diameter and 6 feet high. On top of the cover is a No. 2 Koerting blower which draws the gases from the matting-pot, or from the muffle-furnace, and forces them through the solution in the tank. The lead-pipe in the tank is 4 inches in diameter, and connects with a hexagonal ring, 6 inches above the bottom, into which 1086 holes of $\frac{1}{8}$-inch diameter have been drilled. There is an outlet for the solution with rubber-hose and pinch-cock. The solution can be drawn into a small filter-tank, with Koerting pump below the filter. It is pumped to the wash-water tank. Steam and sulphurous acid escape through a wooden box on top of the cover, and are taken out through the roof of the building.

Dissolving-Tanks.—There are two dissolving-tanks, 3 feet 6 inches in diameter and 5 feet 8 inches high, with conical bottoms. The sides are lined with 8-pound lead, and the conical bottoms with 12-pound lead. At the end of the cone is a discharge-opening, 2 inches in diameter, closed by a tapering, flanged plug of soft rubber. The latter is held in place by an iron thumb-screw. For heating the solution, a 1-inch lead-pipe is provided. It ends in a ring, 12 inches in diameter, perforated with a number of holes, so drilled that the direction of the steam-jets, up and down, is in line with the slope of the cone. For the protection of the lead-lined cone, a false cone of 8-pound lead is inserted round the ring of the steam-pipe. The tanks project 2 feet 6 inches above the charge-floor, and are suspended by flanges of heavy angle-iron, so that their bottoms are entirely free.

Filter-Tanks.—A filter-tank is arranged below each dissolving-tank for receiving the charge from the latter. These tanks are 5 feet 6 inches by 3 feet and 2 feet deep. They are lead-lined and provided with asbestos-cloth filters. The latter are covered with a perforated lead-plate, so that the filter is protected in removing the silver. Each tank has an outlet below the filter, connected with a Koerting pump, made of hard lead.

Crystallizing-Vats.—There are six of these, 6 feet by 3 feet and 2 feet deep, lined with 6-pound lead. The bottom of the vat is covered with an extra piece of sheet-lead. Each vat has an inclined lead-lined drainage-board for receiving the blue-stone crystals.

Wash-Water Tank.—A tank of the same size as the crystallizing-vats is used for receiving weak solutions and wash-water.

Koerting Pumps.—A Koerting pump, made of hard lead, transfers solutions, by means of rubber-hose, from the crystallizing-vats to the dissolving-tanks, etc.

Tanks for Precipitating Cement-Copper.—The two copper precipitating-tanks are 6 feet long, 2 feet wide and 4 feet deep, lined with lead. The bottom is V-shaped. Each tank has a grate on which the scrap-iron rests, and is divided into four compartments, through which the solution is circulated by a small Koerting pump. At the V-shaped ends of the tanks are gates, through which the fine cement-copper, dropping through the grates, can be discharged into a filter-car.

The Cake Press.—For pressing the cement-silver into cakes, a hydraulic press, with mould 6 inches in diameter and 4 inches high, is used. The safety-valve of the force-pump opens when a pressure of 1000 pounds per square inch has been reached. I use pure glycerine in the pump-tank in place of water or alcohol, and find this to answer very well, keeping the pump and valves well lubricated. The press was made by Watson & Stillman, of New York, and works very satisfactorily.

Drier for Silver-Cakes.—A small drier for silver-cakes is heated by air derived from the iron chimney of the muffle-furnace. An air-chamber is formed round the chimney by a mantle of sheet-iron, and from the annular space the hot air is drawn through the drying-chamber by a Koerting ventilator.

Building.—The main building of the refinery is 52 by 42 feet. In an annex, 52 by 25 feet, stand the sulphide-drier, a small engine for driving the ball-mill, cake-press and sample-grinder, a lead-lined tank for storing sulphuric acid, a bin for coal, scales, etc. Here is also room for the storage of sulphides.

The melting of the silver is done in the old retort-room of the mill. The whole building is provided with a smooth floor of cement-concrete on which every speck of dropped sulphides or matte can be seen.

The Process.

Matting.—On account of the large amount of free sulphur contained in the sulphides, they cannot be roasted directly, but must first be converted into a matte. This is done either with or without the addition of copper. In my laboratory-experiments I obtained the best results in roasting by producing a matte with equal percentages of silver and copper; and I followed this practice at first in working on a large scale. But I found later that the addition of copper is not necessary if proper attention is paid to the roasting of the matte.

In matting the sulphides directly, the matte fuses at a very low red-heat, and is exceedingly liquid. The more copper is added, the higher is the temperature at which the matte melts. If the matting is done without incorporating copper, the whole charge is often obtained in liquid form; if a considerable quantity of copper is used, a large portion of the charge, sometimes nearly all of it, comes out in semi-fused, partly oxidized lumps. The liquid matte is poured into a flat box of cast-iron, forming sheets about ¼ inch thick. As soon as the matte solidifies, and before it gets cold, these sheets can be rolled up like sheet-lead. When cold, the matte is exceedingly brittle.

The draught of the matting-furnace is so regulated that the flame playing round the pot is smoky and reducing. A bright, oxidizing fire would soon destroy the pot by forming burnt scales on the outside. The cast-iron is not attacked by the liquid matte on account of the great affinity of copper for sulphur. Small pieces of wrought-iron, derived from the cement-copper, which occasionally get into the charge, remain intact, and are found in the Brueckner pulverizer when a clean-up is made. The iron ladles, spades, and pokers used in matting do not show the least wear after considerable service.

About 800 pounds of sulphides are matted in one shift, consuming about 220 pounds of coal.

Pulverizing the Matte.—The matte is allowed to get perfectly cold before pulverizing. The colder it is the quicker it pulverizes. When the Brueckner ball-mill gets warm, it is best to allow the machine to cool down. The product, pulverized through a No. 40 screen, is scaly and granular, and absolutely free from dust. If the pulverizer is to be cleaned, the charge-hopper is removed, and the balls are taken out by hand through the charge-opening. Pieces of unpulverized matte are scooped out, one of the screens is removed, and the machine is revolved for a few minutes until the small pieces of matte between the cylinder and the screens have dropped out. The pulverizer is then ready for grinding roasted matte.

Roasting.—The muffle-furnace handles conveniently a charge of 600 pounds of matte. One-half the charge is generally put in at once, and the other half added gradually after the matte has caught fire. It is essential that the muffle should have the right temperature. If it is too cold, the charge will become like mush, and in such a case it is best to draw it out and get the furnace hot. The temperature should be sufficiently high to ignite the matte about fifteen or twenty minutes after charging. Two men, one at each door, keep the charge in constant motion, working it alternately with rake, spade, and hoe. The draught in the muffle should be so regulated that fumes do not escape through the working-doors. If necessary, the solution in the Roessler tank is lowered, whereby the amount of air drawn by the Koerting blower is materially increased.

During the period of desulphurizing, very little fire is needed. Later on, the heat is increased. It takes generally eight hours to finish a charge. The charge is drawn when a sample, dropped into a beaker filled with hot water, looks black, shows a weak spangle-reaction (precipitation of metallic silver from silver sulphate by cuprous oxide), and the solution gives a copious precipitate of silver chloride by hydrochloric acid. Small lumps, when broken, should be without a red kernel of cuprous oxide. The charge is dropped from the car into a flat cast-iron pan, and allowed to cool there slowly over night. Large lumps, which come principally from the sides of the muffle in cleaning the hearth after discharging, are raked out and re-crushed with the raw matte. The same is done with scales from tools. The cooled roasted matte is accumulated in iron chests, and finally pulverized through a No. 40 screen in the Brueckner mill.

During the eight hours' roasting about 1000 pounds of coal are consumed.

The roasting, as conducted at the Marsac mill, can hardly be called a Ziervogel roasting. In fact, it is not desirable to form a large percentage of silver sulphate. If a sample of the pulverized roasted matte is dissolved in dilute sulphuric acid, the solution shows rarely a reaction for silver, the cuprous oxide present decomposing the silver sulphate completely.

In the iron pipe connecting the muffle with the Roessler, about a handful of scales is found after roasting a charge. They consist mostly of soluble salts, and contain from 300 to 400 ounces of silver per ton. This would indicate that the volatilization of silver in roasting is very slight. The precipitate found in the Roessler assays from 470 to 900 ounces in silver per ton, and nearly all its silver is soluble in extra-solution, indicating that the silver is present as sulphide. Its quantity has not yet been determined, owing to the difficulty of filtering the solutions drawn from the Roessler. The turbid solution goes to the copper precipitating-tanks, and here the fine precipitate adheres to, and settles with, the cement-copper, and is not lost.

Dissolving.—A charge consists of 16 cubic feet of mother-solution from the crystallizing-vats, the necessary sulphuric acid, and 300 pounds of pulverized roasted matte. After determining the precentage of copper in the matte and the free acid in the mother-solution (by titration with a solution of sodium carbonate), sufficient 66° B. acid is added to have 2 pounds of H_2SO_4 for each pound of copper contained in the matte. Steam is now turned on, and the solution is heated to boiling. The roasted matte is then slowly charged with a hand-shovel, the solution being kept in constant ebullition by the steam. After charging, boiling is continued for two hours. Copper-plates are suspended in the solution to precipitate silver from any silver sulphate not decomposed by cuprous oxide. The wear of these plates is, however, very slight. Discharging the solution is effected by removing the thumb-screw below the rubber-plug, and punching out the latter with a wooden rod from above. A lead-cylinder prevents spattering of the solution. Silver remaining in the cone is washed out with a bucketful of hot solution.

Filtering.—The filter-tanks are kept covered, so that the solution remains hot. The Koerting pump below the filter is not turned on in the beginning, but only after nearly all the solution has filtered through. It is necessary to remove the concentrated solution below the filter to prevent crystallization on cooling. By rubber-hose and lead-lined troughs, the solution is conveyed to the crystallizing-vats. It takes from one and one-half to two hours to filter a charge. The solution is perfectly clear if the filter is intact. A defective place is easily repaired by placing a piece of asbestos-cloth over the rent after raising the perforated lead-plate. The concentration of the solutions is generally from 42° to 43° B. Solutions of much higher concentration do not filter well.

If a drop of hydrochloric acid is added to the hot solution, a copious white precipitate appears, which dissolves completely upon the addition of an excess of acid. This is cuprous chloride, showing the presence of cuprous sulphate in the solution. In cooling, the cuprous sulphate decomposes, forming cupric sulphate and metallic copper; the cold solution does not give a precipitate with hydrochloric acid. A thin coating of bright-red metallic copper is found at the bottom of the crystallizing-vats and on the strips of lead suspended in the vats for crystallizing blue-stone.

If fine blue-stone crystals are mixed with the metallic copper, their color becomes almost white, red and blue being complementary colors. One might assume, from appearances, that these white crusts contained silver. But in dissolving them a bright-red and exceedingly fine powder of metallic copper appears on the bottom of the beaker.

Washing the Silver.—After six charges have accumulated in a filter tank, washing is commenced. About six inches of water are run on the top of the silver, and this is circulated by the Koerting pump until the solution is hot, and shows no change in specific gravity. The latter is generally from 20 to 22° B. Now, the solution is discharged into one of the crystallizing-vats. Washing with fresh water is continued, and the wash-water is run to the weak-solution tank until it is so weak that it shows only a fraction of a Baumé degree. At this point the wash-water is turned into a tank filled with scrap-iron, outside of the building. Washing is suspended when the water shows only a very faint blue color upon adding a few drops of ammonia.

Pressing the Silver.—The washed silver is rather too moist for convenient pressing. It is put in the drier and left there until most of the water has evaporated.

The cakes from the press weigh from 7 to 9 pounds, and are put back in the drier, and then melted.

Blue-Stone.—The crystallization of the blue-stone is very rapid, and is hastened by suspending 48 strips of lead, 3 inches wide and 20 inches long, in each vat. Since most of the blue-stone is consumed in the mill, a showy article is not required, and small crystals dissolve much better than large ones in making extra-solution.

When the mother-solution has gone down to about 30° B., it is used in the dissolving-tanks. The comparatively high specific gravity of the mother-solution is partly caused by the presence of about 8 per cent. of free H_2SO_4.

The volume of the solution does not increase.

After prolonged use, the mother-solution accumulates iron vitriol and other impurities. When this is the case, it is pumped to the copper precipitating-tanks.

Precipitation of Cement-Copper.—As already stated, impure mother-solutions, solutions from the Roessler, and wash-water from the filter-tanks, are precipitated with scrap-iron. After a number of charges have been put through, the cement-copper is washed twice with water before making a clean-up. The fine copper below the grates is drawn into a filter-car and washed there. It generally contains more or less silver derived from the turbid Roessler solution, and also from silver-bearing scales on worn-out solution-pipes, used for precipitation of the copper. All cement-copper goes back to the matting-furnace.

Economical Results of the Process at the Marsac Refinery for the Year 1892.[*] [†]

Before starting on the work of the year 1892, a most careful clean-up was made, and any remnants were charged to the refinery. During 1892 the Marsac lixiviation-works produced 48.8 tons of so-called "regular" sulphides, which contained, on an average, 11,449 ounces of silver per ton; 11.77 ounces of gold per ton; and 569.8 pounds of copper per ton. To this must be added, 266 pounds of sulphides and by-products left over from the product of 1891, which contained 1380 ounces of silver, 1.41 ounces of gold, and 63 pounds of copper. Before giving the results of the clean-up, a general discussion of the items of expense, methods of assaying, etc., is desirable.

---

[*] Trans. Am. Inst., Vol. XXIV.

[†] After the clean-up for the year 1892 had been made, it was discovered that silver had been stolen by one of the laborers in the refinery. The quantity traced, and all shipped and sold to one firm of jewellers in Chicago, amounted to 1821.75 ounces of silver, and 1.87 ounces of gold, *i. e.*, it represented over three-tenths of one per cent. of the whole output. While thus the apparent loss of silver was considerably reduced, it is doubtful whether the remaining loss is not also in part due to stealing, so that the process may have done metallurgi-

Labor.—The 48.93 tons of sulphides (and remnants) were put through the refinery in 10 months and 20 days, requiring in a day-shift the labor of two men (one at $4.00, one at $3.50), and one boy (at $2.00). This is exclusive of the labor for melting bullion.

Coal.—Of the coal delivered at the refinery, a portion was consumed in the stove supplying hot air for the sulphide-drier. Mr. Lamb estimates that this was fully one-quarter of the quantity delivered, and I have made a deduction accordingly.

Steam.—The cost of steam consumed by Koerting pumps and blowers, for running the ball-mill and cake-press and for heating solutions in the dissolving-tanks, is estimated by Mr. Lamb at $929, and I consider this rather too high than too low.

Cement-Copper.—It will be noticed that the refinery produced about as much cement-copper as it consumed; hence, the cement-copper does not enter as an item of expense. The iron consumed in the precipitation of copper costs nothing, there being always on hand at the mill an abundance of old wrought-iron. The cement-copper results by precipitation of wash-water from the silver filter-tanks, and of impure mother-solutions from the crystallizing-vats, *i. e.*, the latter solutions go to precipitation as soon as a mixed copper and iron vitriol commences to crystallize.

Bullion-Melting.—The melting of the silver-cakes was done in crucibles; it should have been done in a reverberatory-furnace. Such a furnace was introduced by the writer many years ago at the Ontario mill. Since at the Marsac mill no accurate account

cally and financially much better work than is represented here. The stolen silver has been introduced in the statistical figures as a separate item, wherever necessary. It does not influence the expense-account, except express and refining charges, and these are deducted from the value of this silver in figuring the net amount of cash realized. It is not necessary to introduce the item of reclamation—no doubt, the Chicago jewellers who buy stolen silver have already taken care of that.

was kept of the cost of crucible-melting, I substitute here the cost of reverberatory-furnace melting.

According to A. D. Moffat, 1449 bars, weighing 2,152,523 ounces troy, or 147,635 pounds avoird., were melted at the Ontario mill, during 1892, consuming materials and labor as follows:

36 cords of wood, at $5.00	$180 00
3240 pounds of borax, at 9 cents	291 60
Labor for melting and weighing	270 00
For 2 new hearths:	
1 mason for 5 days, at $5.00	25 00
200 Denver fire-brick	12 00
100 pounds of fire-clay	1 00
Blacksmith's work, repairing tools	10 00
Total	$789 60

This would be 36.68 cents per 1000 ounces of bullion in bars.

The 36 cords of wood could be easily replaced by 24 tons of Coalville coal, at $4 per ton, reducing the cost of fuel to $96, and the total cost to $705.60, or, the melting of 1000 ounces of bullion to 32.78 cents.

About 2 tons or more of bullion are accumulated before melting is started, and this quantity is put through in three charges. The night-watchman starts the fire at 1 A. M., so that the first melt is ready to draw at 7 A. M., when the two melters go on shift. The furnace once hot, the two other melts follow quickly, and by 3 P. M. the whole work is done, including cleaning, dressing and weighing of the bars. After drawing slag, the metal is well stirred and tapped into a row of moulds resting on a movable car. Only one assay is necessary for the bars belonging to one melt. The hearth of the furnace is made of the best Denver fire-brick, and lasts for about 54 melts before renewal becomes necessary. The arch over the hearth of the furnace is movable so that repairs can be easily made. I do not know of a more simple and economical method of melting bullion. The first furnace of this kind was built by the writer, in 1882, at the Lexington mill, Montana. The flue of the furnace should be connected with a dust-chamber for the deposition of fine silver-globules carried forward by the draught during the boiling of the metal.

Refinery-Cleanings.—The large quantity of refinery-cleanings

(6.5 tons) consisted principally of ashes, unburnt coke, slags, and broken crucibles from bullion-melting, and some matte-soaked bricks from the muffle-furnace. The crucibles were of poor quality, an old lot being on hand in the store-house, originally made for melting antimony. With reverberatory-furnace melting of the silver-cakes these refinery-cleanings would be very small in weight and carry much less silver.

Assays.—The mill-assays of the sulphides represent the so-called "corrected" assays, *i. e.*, to the precious metals found by a careful scorification-assay, the silver and gold contained in the slag of the scorifiers and in the cupels are added, which increases the original result from 1.2 to 1.3 per cent.

Sale of Refinery-Cleanings.—The 6.5 tons of refinery-cleanings were sold to the Omaha-Grant smelter at sulphide-rates (these will be stated later on). A settlement was made on the basis of 7400.45 ounces of silver and 5.223 ounces of gold, according to mill-assays. At the time of the sale the price of silver was lower than 84.864 cents; but having assumed this price throughout, this being the average price for which silver in bars was sold, I retain it here for the sake of uniformity in making comparisons. On this basis, the silver in refinery-cleanings brought $6091.91, and the gold, $104.46, total, $6196.37, less freight and charges of $30 per ton.

Returns of the Aurora Refinery.—The Marsac bars, on an average .927 fine in silver and .00096 fine in gold, were all shipped to the Aurora refinery. The latter first remelts the bars in large crucibles, and then charges 0.7 cent per gross ounce on their weights after remelting, for parting and refining. The refined bars were then shipped for sale to New York. The following table shows the differences between the mill-assays and the returns of the Aurora refinery.

Table Showing Reclamations of Aurora Refinery.

	Fine Silver. Ounces.	Fine gold. Ounces.
Marsac assays	545,044	569.752
Aurora returns	543,003	536.563
Reclamations	2,041	33.189
Or	0.37 per cent.	5.82 per cent.

The average price at which the silver was sold in New York was 84.864 cents per fine ounce. The whole product brought:

For silver	$460,814.80
For gold	11,090.74
Total	$471,905.54

The reclamations amounted to :

On silver	$1732.07
On gold	686.00
Total	$2418.07

It will be seen that the Aurora reclamations on silver are not excessive, while those on gold are out of proportion. So far as the extraction of gold in the Marsac refinery is concerned, there is no reason why it should be below that of silver ; on the contrary, it should be in excess, as shown by all the Marsac mill-assays. The Aurora returns, however, make the apparent loss in gold very large.

STATISTICS OF THE MARSAC REFINERY.

The following statement shows the quantities of metals and sulphuric acid sent to the refinery :

Sent to the Refinery.

	Ounces.
Silver, in sulphides	558,715
" in remnants	1,380
Total	560,095
Gold, in sulphides	574.42
" in remnants	1.41
Total	575.83
	Pounds.
Copper, in sulphides	27,805
" in remnants	63
" in cement-copper	3,363
Total	31,231
Sulphuric acid (66° B)	52,906

This gives a consumption of 0.54 of a pound of sulphuric acid per pound of sulphides, or 1.74 pounds per pound of copper actually dissolved, or 1.96 pounds per pound of copper in blue-stone.*

The comparatively low consumption of sulphuric acid, considering that, besides the copper, some impurities in the roasted matte are also dissolved, is explained by the fact that the roasted matte contains more or less silver sulphate and cuprous oxide, which react upon each other in the dissolving tanks, precipitating silver and forming copper sulphate. On account of the presence of cuprous oxide, the copper-plates suspended in the dissolving-tanks are not perceptibly attacked, and are kept there only as a matter of precaution.

---

* These figures are not absolutely correct, but as close as possible under the circumstances. A few hundred, or even a thousand pounds more or less of sulphuric acid would not influence the expense-account materially.

Product of the Refinery.

(Mill-assays).

Silver :*	Ounces.	Per cent.	
In bars	545,044	97.313	97.638
Stolen	1,822	0.325	
Refinery-cleanings	7,400	1.321	
Final cleanings	169	0.030	
Not accounted for	5,660	1.011	
Total	560,095	100.000	

Gold :*	Ounces.	Per cent.	
In bars	569.752	98.945	99.270
Stolen	1.870	0.325	
Refinery-cleanings	5.223	0.907	
Final cleanings	0.071	0.012	
Total	576.916	100.189	
Plus of gold	1.086	0.189	

Copper :	Pounds of Copper.
In 108,011 pounds of blue-stone	27,003
In cement-copper	3,404
In refinery-cleanings	509
Not accounted for†	315
Total	31,231

* In figuring the product on the basis of Aurora returns, the silver in bars would be 96.948 per cent., and the loss 1.376 per cent. ; the gold in bars would be only 93.180 per cent., and the loss as large as 5.576 per cent. As already remarked, there is no reason why the loss in gold should be so much in excess of the loss in silver, and the Aurora gold returns are manifestly too low. The assay of sulphides for gold is liable to be below the actual contents in gold ; hence the plus of gold.

† The copper not accounted for is in the silver bars. The latter were not tested for their percentage in copper. If all the base metal in bars were copper there would be an excess of the latter in the clean-up. On the other hand, more mother-solution may have gone to precipitation than actually belonged to the clean-up for 1892. Since, however, it was found that the addition of cement-

Of the blue-stone, there was sold:

To outsiders, 11,637 pounds, at 6¼ cents	$ 726 30
To the mill, 96,011 pounds, at 5½ cents	5,300 57
Total	$6,026 87

Expenses of the Refinery.

Labor	$3,040 00
Coal, 170 tons, at $4.00	680 00
Sulphuric acid, 52,906 pounds, at 2.48 cents	1,312 07
Materials and supplies	269 00
Melting bullion	192 00
Steam and power	929 00
Freight on bullion*	5,314 77
Refining and commission†	4,397 81
Sacking refinery-cleanings	65 00
Freight and charges on the same	195 00
Total	$16,394 65
Less blue-stone sold	6,026 87
Total, net	$10,367 78

Or 1.851 cents per ounce of fine silver in sulphides.

The silver-products of the Marsac refinery were sold as follows:

Silver-Products Sold.

Bars	$471,905 54
Value of stolen bullion, net	1,543 92
Refinery-cleanings	6,196 37
Value of final cleanings	145 30
Total	$479,791 13
Less expenses	10,367 78
Realized net	$469,423 35

copper in matting sulphides is not essential to the success of the process (as had been assumed in the beginning), it is not necessary to buy cement-copper, but only to work in the cement-copper produced in the refinery. This copper always contains a small amount of silver, derived partly from the turbid solution coming from the Roessler tank, and partly from silver-bearing scales on old wrought-iron used for the precipitation of copper.

* Freight on bars from Park City to New York, *via* Aurora, is 0.975 of a cent per fine ounce of silver, as reported by mill-assay.

† The broker's commission for selling bullion in New York is $\frac{1}{16}$ of one per cent. on value. This and the Aurora refining-charges make together a cost of 0.806 of a cent per fine ounce of silver, on the basis of mill-assay.

Sale of Sulphides to Smelters.

Before the Marsac mill erected its own refinery the "regular" sulphides were sold to the Omaha-Grant Company on the following terms: freight and charges, $30 per ton; 97 per cent. of the silver paid for at New York quotations; $20 per fine ounce of gold. To this must be added the cost of sacking sulphides and hauling them to the Park City R. R. depot, at $10 per ton, and the reclamations on the mill-assays. It will be of interest and importance to investigate the differences between the mill-assays and the smelters' assays of sulphides.

The sulphides are first carefully sampled and assayed at the mill, and then put up in double sacks for shipment. After arriving at the smelter they are resampled and assayed, and a bottle is sent to the mill for assaying. The two samples and three assays form the basis of settlement between the mill and the smelter, but there are no fixed rules about it. In most cases a splitting of differences is resorted to.

The following abbreviations will here be used: M. S.—M. A., for mill-sample and mill-assay; O. S.—M. A., for Omaha and Grant sample and mill-assay; O. S.—O. A., for Omaha and Grant sample and assay.

On eight lots of "regular" sulphides, sold in 1890, weighing 17.5 tons, and containing, as per M. S.—M. A., 218,324 ounces of silver and 127.63 ounces of gold, the differences in assays were as follows:

Comparing O. S.—M. A. with O. S.—O. A., the latter was 0.587 per cent. lower in silver and 3.787 per cent. lower in gold than the former.

Comparing M. S.—M. A. with O. S.—O. A., the latter was 1.419 per cent. lower in silver and 5.469 per cent. lower in gold than the former.

In effecting a settlement, a deduction or reclamation of 0.564 per cent. in silver and of 1.802 per cent. in gold, based on M. S.—M. A., was accepted by the mill.

On five lots of "regular" sulphides, sold in 1891, weight 16.72 tons, and containing, as per M. S.—M. A., 156,091 ounces of silver and 134.28 ounces of gold, the differences in assays were as follows:

Comparing O. S.—M. A. with O. S.—O. A., the latter was 1.355 per cent. lower in silver, but 2.271 per cent. higher in gold than the former.

Comparing M. S.—M. A. with O. S.—O. A., the latter was 2.020 per cent. lower in silver, but 3.951 per cent. higher in gold than the former.

Separate terms of settlement for these lots I did not obtain; they are included in the twenty-six lots following.

On twenty-six lots of "regular" sulphides, base sulphides, and carbonates, sold in 1891, weight 76.69 tons, and containing, as per M. S.—M. A., 505,255 ounces of silver, and 427.28 ounces of gold, the difference in assays were as follows:

Comparing M. S.—M. A. with O. S.—O. A., the latter was 1.923 per cent. lower in silver, but 3.796 per cent. higher in gold than the former.

In effecting a settlement, a reclamation on silver of 0.931 per cent. was accepted by the mill, and a gain in gold of 1.633 per cent. was conceded by the smelter.

Outside of these twenty-six lots, twelve lots of sulphides were sold in 1891, on which no reclamations were made.

On the total production of sulphides, sold in 1891, there was a reclamation of 0.616 per cent. in silver and a gain in gold of 1.078 per cent. on the basis of M. S.—M. A.

Finally, I received from the Daly office a statement of reclamations on all lots of "regular" sulphides sold in 1889, and up to 1892. These contained 1,774,776 ounces of silver and 1474.51 ounces of gold. The reclamations on silver were 5100 ounces, or 0.287 per cent., and on gold 4.60 ounces, or 0.312 per cent.

It will be seen from the above figures, that there is more or less, and rather variable, discrepancy between the respective samples and their assays. This is not surprising, in consideration of the richness of the material, the different methods of assaying, and the effort of the smelter not to be found on the losing side. It is not necessary to make further comments on these figures—they tell their own story.

Net Profit Realized by Selling Sulphides to Smelters.

It now remains to investigate how much the mill would have realized by selling the product of "regular" sulphides for 1892 to smelters. It is somewhat difficult to decide on which basis of reclamation this is to be done, and Mr. R. C. Chambers very properly remarks in this respect, "There are no figures that can be furnished giving a reliable guide as to what would be the loss through reclamations in the future, because there has been no regularity in the same."

Under the circumstances, I consider it best to use the last statement of reclamations, which is the most unfavorable to the refinery. In this statement, the reclamations on silver were 0.287 per cent., and on gold 0.312 per cent. Hence, we have to deduct from the silver-product of 560,095 ounces for 1892, first, 0.287 per cent. reclamations, and then, 3 per cent. smelters' discount, which leaves 541,734 ounces of silver at 84.864 cents, or $459,727.14. From the gold product of 575.83 ounces we have to deduct 0.312 per cent. reclamations, which leaves 574.04 ounces at $20, or $11,480.80. From this we must deduct $1957.20 for sacking, and for freight, and smelters' charges on 48.93 tons of sulphides.

Realized by Selling to Smelters.

Silver sold for	$459,727 14
Gold sold for	11,480 80
Total	$471,207 94
Less Expenses	1,957 20
Realized net	$469,250 74
Realized net by refinery	469,423 35
Difference in favor of refinery	$172 61

Using, however, as a basis for calculation, the reclamations on the total product of sulphides sold in 1891—which seems to me more just and equitable, because these reclamations refer to the last sales—the difference between selling to smelters and refining at the mill by the old process would have been $1520.28 in favor of the refinery.

From this it appears, that in order to make refining at the mill *decidedly* profitable, the process, as conducted in 1892, must be improved; and this point will be discussed in my concluding remarks. In making this statement it is assumed that the loss of 1 per cent. in silver is a *bona fide* loss, and was not caused by further stealing.

Finally, it can be easily seen that, with a lower price for silver, the profit in refining at the mill, as compared with selling to smelters, becomes less, provided the terms of the latter remain the same. The expenses of refining are independent of the price of silver, while the smelter's discount of 3 per cent. diminishes in dollars and cents with a drop in the price of silver. It is evident, however, that when a certain limit has been reached, the smelter will have to raise his deductions if he wants to make the same profit as formerly. It is, really, the deductions and not the freight and charges of $30 per ton, out of which his profit is derived. With each decline in the price of silver of 1 cent per fine ounce, the smelter's margin of 3 per cent. on sulphides containing 11,449 ounces of silver per ton, declines $3.43. Considering the recent drop in the price of silver, this lessening of the smelter's profits becomes quite serious.*

Conclusion.

This process for refining sulphides, which worked very smoothly in a technical sense, can be made more economical in expense and less wasteful in silver. The plant, as now constructed, requires only an addition of crystallizing-vats for blue-stone to handle more than double the quantity of sulphides, which is important in view of the erection of lixiviation-works by the Ontario mill.

The muffle-furnace, which was running intermittently, proved, in consequence, to be very wasteful in fuel. It took more coal to heat up the furnace in starting roasting, and to keep it hot for 16 hours, while no roasting was done, than the quantity consumed during roasting itself. Nor would it be necessary to employ twice the number of men in doubling the capacity of the refinery

The selling of sulphides to smelters should be avoided, if possible. There is too much annoyance and uncertainty in effecting a just and equitable settlement. Every improvement in the present plant, and in the system of working, will materially increase the balance in favor of refining at the mill.

The most important point for consideration is, to reduce the loss of silver. Where does this loss take place? If the products are carefully handled, the mechanical loss by dusting is very slight. The gases from both the matting-furnace and the muffle-furnace pass through the Roessler converter, where dust and fumes are arrested. In boiling the roasted matte in the dissolving-tanks some fine silver may be carried forward by the steam; and these tanks should be covered with lead-lined hoods. If the asbestos filters in the silver-tanks are carefully watched, there is little danger of silver escaping with the copper-solution. I do not hesitate to say, that by far the largest part of the silver-loss was due to the present method of melting the silver-cakes in crucibles. The silver-cakes contain more or less impurities, which cause the metal to boil during and after fusion. In fact, the silver globules adhering to the sides of the melting-furnace, and accumulating in the flue leading to the chimney, and the large amount of silver found in the ashes, are sufficient evidence in support of this statement. Reverberatory-furnace melting would reduce these losses to a minimum.

---

* This prediction has been verified. On a lot of regular sulphides, sold by the Marsac mill, in 1893, the reclamations of the smelter were unusually heavy.

It would be well to moisten the cement-silver before pressing into cakes with a concentrated solution of borax, perhaps with addition of nitre. A thorough drying of the cakes would not be necessary. The reverberatory-furnace could be charged with them, and a slow fire started to drive out all moisture first. This method of melting would also reduce the weight of refinery-cleanings to a minimum, thus causing less expense in their reduction.

A further saving could be made by the introduction of an electrolytic plant for parting and refining the *doré* bars. This would do away with the Aurora reclamations of $2418. The metal from the melting-furnace could be cast at once into plates for the electrolytic refinery. It is claimed that the cost of refining *doré* bars in the Moebius electrolytic plant at Mansfield, Pa., is only $\frac{1}{8}$ of 1 cent per ounce of bullion. If this be true, I should think $\frac{1}{2}$ cent would be a liberal charge at Park City, provided the dynamos were run by water-power from the Ontario tunnel.

NOTE.—In 1893 the above-described process was abandoned at the Marsac mill, the Dewey-Walter Refining Co. taking a contract for refining the "regular" sulphides at a charge of 1¼ cent per ounce of fine silver, guaranteeing to return the precious metals in the shape of fine bars as per "corrected" assays of the sulphides, but retaining the by-product of blue-stone.

§ 1703. *The Sulphuric Acid Process for Refining Sulphides.*

In this process the sulphides are dissolved, in cast-iron pots, by concentrated, boiling sulphuric acid. The solution is then separated from the gold-bearing insoluble residue, and the silver precipitated as cement-silver with copper. The apparatus and manipulations of the process are similar to the refining and parting of low-grade *doré* bullion. The fumes from the pots, however, contain volatilized sulphur, which carries forward some of the silver, and the gold-bearing residues are comparatively low in gold, but high in silver. They may also contain considerable lead sulphate and other impurities. It is not feasible to treat these residues at the mill—they are sold to smelters.

The Sulphuric Acid Process at the Aspen Mill.

The process was first introduced by Dr. Godshall at Aspen, but used there for a short time only and then abandoned, because it proved to be unprofitable as compared with selling sulphides to smelters.

Below are the statistics on a run of 15.903 tons of sulphides containing, on an average, 3,602.28 ounces silver and 1.272 ounces gold, per ton, besides 35.2 per cent. copper, 5.4 per cent. lead, 20.9 per cent. sulphur, and from 0.4 to 13.6 per cent. CaO.

Product.

Silver:

53 bars, of 986.09 average fineness, and containing 46,505.46 ounces fine silver.

Residues, 8011 pounds, containing:

9,476.95	ounces	silver.
20.3098	"	gold.
2026	pounds	lead.

By-Products:

Condensed fumes from pots, 327 pounds, with	50.00	ounces	silver.
Slag from melting-furnace, 144 pounds, with	79.41	"	"
Ashes from melting-furnace, 690 pounds, with	37.67	"	"
Granulated silver,	206.00	"	"
Cement-copper, 2226 pounds, with	73.20	"	"
Total:	450.28	ounces	silver.

Blue-Stone, 37,350 pounds.

The silver was distributed as follows:

In:	Per cent.
Bars	81.178
Residues	16.543
By-Products	0.786
Not accounted for	1.493
	100.000

All the gold was recovered in the residues with a fraction of an ounce over.

The expense of refining—labor, coal, coke, acid, repairs, etc.—including cost of disposing of residues and by-products, express-charges on bars, etc., and deducting value of blue-stone, was $276.24 per ton of sulphides, or 7.66 cents per ounce of fine silver in sulphides. The greatest expense was for sulphuric acid (66° B.) of which 4.01 pounds were consumed per pound of sulphides, costing 8.421 cents, or $168.42 per ton of sulphides.

If the sulphides had been sold to smelters, the cost per ton, including freight and deductions, would have been $199.47, or 5.554 cents per ounce of fine silver in sulphides, a difference of $76.77 per ton, not counting the value of the silver lost, or not accounted for, in refining.

It is evident that this process must be unprofitable on sulphides of such low grade in silver as those produced at Aspen.

The Sulphuric Acid Process at the Marsac Mill.

As already stated, the sulphuric acid process was introduced at the Marsac mill under contract by the Dewey-Walter Refining Company. The latter works under U. S. patent issued to F. P. Dewey.

Mr. Dewey has kindly supplied the following statistics concerning the refining of "regular" sulphides produced in 1894.

Sulphides treated: 116,519.5 pounds.

The sulphides contained, by corrected assay:

Silver	572,544.4	ounces
Copper	31,585.3	pounds
Lead	385.6	"

(Gold-assay not given.)

Hence, the average assays were as follows:

Silver	9,827.4	ounces per ton.
Copper	27.1	per cent.
Lead	0.33	"

The following materials were consumed:

Sulphuric acid (66° B.)	389,439	pounds
per pound of sulphides	3.34	"
per ounce of fine silver	0.68	"
Coal	166,000	"
Copper to precipitate silver	16,832.5	"
One pound copper precipitated silver	2.27	"

The fuel, crucibles and flux for melting bullion are not included in the above statement of materials consumed.

Labor: One foreman and two men, per day.

Extra labor: One man melting bullion; laborers for making clean-up and mechanics for making repairs, etc., whenever necessary.

401 silver-bars, free from gold, were produced, .999½ fine, and 45 bars, .999 fine.

The silver in the products returned was distributed as follows:

	Per cent.
Fine bullion in bars	96.29
In gold-bearing residues	2.76
In cleanings	0.93
Silver on hand	0.38
	100.36

This makes a plus clean-up of 2,078.8 ounces of silver.

175,809 pounds of blue-stone were produced, or 3.63 pounds of blue-stone per pound of copper gone to the refinery in sulphides and as metallic copper for precipitating silver.

The gold-bearing residues were shipped and sold to smelters. Mr. Dewey contemplates, however, to refine the residues at the mill. There was an apparent loss in gold, the cause of which has not yet been fully investigated. Since a plus clean-up was made in silver, it appears strange that gold should have been lost, provided the silver-bars were entirely free from gold.

CHAPTER XVIII.

The Russell Process Compared with Ordinary Lixiviation, and with Amalgamation.

A. The Russell Process Compared with Ordinary Lixiviation.

§ 1801. *Results at the Cusihuiriachic Mill, Chihuahua, Mexico.*

Special mill-runs were made to determine the difference in extraction between ordinary lixiviation and the Russell process, *i. e.*, the roasted ore was treated by the old process until the tailings could not be further reduced in value, and then the extra-solution was applied. In thus treating 92 charges = 803 tons of ore, containing on an average 43 ounces of silver per ton, the following results were obtained :

	Per cent. of Silver extracted in :
Assay-office by ordinary solution	78.6
" " extra-solution	88.3
Mill by ordinary process	60.7
" Russell process	83.3

In regular mill-work, using the Russell process, the time of treating a charge was reduced from 66 to 41 hours, compared with ordinary lixiviation. A notable discrepancy exists in this experiment between the assay-office tests and the mill-extraction, especially concerning the ordinary solution. This is caused by the chlorinations "going back" after the first wash-water has been applied. Howell furnaces were used for the chloridizing-roasting of Cusi ore.

§ 1802. *Results at the Sierra Grande Mill, New Mexico.*

The Sierra Grande ore contained lime, and the first wash-water, applied to the roasted ore, showed a decided alkaline reaction. The chloridizing-roasting of the ore was conducted in White furnaces without auxiliary fire-places for roasting the dust. Below are the average results running the mill for 7 months on ore assaying from 17 to 12 ounces of silver per ton.

Per cent. of Silver extracted in:	
Assay-office by ordinary solution	71.0
" " extra-solution	81.5
Mill by ordinary process	53.0
" Russell process	83.0

We note here again a considerable "going back" of chlorinations in the mill.

§ 1803. *Results at the Yedras Mill, Sinaloa, Mexico.*

The statements in this paragraph are taken from a paper on the Russell process* by R. F. Letts, late superintendent of the mill.

The following analyses of Yedras ore, represent its composition at different periods:

	No. 1.	No. 2.
Carbonate of lime	33.78	46.50
Silica	15.13	25.00
Iron	17.33	9.80
Sulphur	13.31	12.50
Arsenic	9.82	2.50
Zinc	4.92	----
Lead	1.78	----
Magnesia	2.58	----
Alumina	1.35	----

No. 2 is an analysis of an average battery-sample for one month. The composition of the ore varies greatly. A couple of months after analysis No. 2 was made, the battery-samples for several weeks had 4 per cent. zinc, and two months later they contained great quantities of antimony.

The value of the ore in silver is about 60 ounces, more or less, per ton.

The Russell process was introduced at the Yedras mill (Anglo-Mexican Mining Company) in November, 1887, and has been in continuous use since then, having superseded the old lixiviation-process. The ore is crushed by stamps through a No. 22 screen, and is roasted with 6½ to 7 per cent. salt in long reverberatory-furnaces at moderate heat. The roasting is conducted with a view of reducing the loss of silver by volatilization to a minimum, *i. e.*, from 6 to 8 per cent. While longer continued roasting at a higher temperature would increase the percentage of silver extracted by lixiviation from the roasted ore, the gain would be more than balanced by increased loss of silver during roasting.

---

* Published by the Russell Process Company, 1893.

The capacity of the plant is 60 tons per day.

Wash-water and hyposulphite solutions are used cold. The strength of the stock-solution in sodium hyposulphite averaged during the years 1888, '89 and '90, 1.89, 1.43 and 1.40 per cent. respectively. The rate of lixiviation was low, namely, from 3.34 to 4.38 inches.

All the following statements about volumes of water or solutions are figured per ton of ore.

From 21 to 23 cub. ft. first wash-water were consumed. This was followed by ordinary solution of which, during the years mentioned above, 51.3, 50.3 and 86.0 cub. ft. were used respectively. The 13 cub. ft. standard extra-solution, which now follow, were allowed to stand for 12 hours in the charge, and were not circulated. The extra-solution was then replaced and followed by 114.1, 94.3 and 140.0 cub. ft. ordinary solution respectively. The volume of the second wash-water was 13 cub. ft. The total time consumed in treating a charge of 16 to 18½ tons of ore, having a depth of 53 inches after leaching with water, was 116.9 hours in 1888; 132.9 hours in '89; and 137.8 hours in '90.

Lead is not precipitated separately at Yedras, only a very small amount of this metal being present in the ore and going into solution.

The average consumption of chemicals per ton of ore during 5 years, from 1888 to '92, was as follows:

Sodium hyposulphite	0.96	pounds.
Blue-stone	5.73	"
Caustic soda	3.76	"
Sulphur	2.79	"

The highest consumption of blue-stone was in 1888, namely, 7.02 pounds; and the lowest in '90, namely, 3.85 pounds. The other chemicals varied very nearly in proportion to the amount of blue-stone used.

Various tests made during the years 1888 to '92 have shown that the use of extra-solution in lixiviating Yedras ore increased the extraction of silver on an average 16.46 per cent.; and the net profit derived from the Russell process, after deducting extra-cost of chemicals, has been from $6.00 to $7.00 per ton of ore, as compared with the old process.

The Yedras sulphides are abnormally high in value. During 1890, with separate precipitation of the extra-solution, the wash-water sulphides contained 46.47 per cent.; the sulphides from ordinary solution 69.1 per cent.; and those from extra-solution 24.79 per cent. of silver. This would correspond to 13,550 ounces, 19,149 ounces, and 7228 ounces of silver, per ton, respectively.

The silver in these precipitates was distributed as follows: Wash-water product, 10 per cent.; in sulphides from ordinary solution, 75 per cent.; in extra-solution sulphides, 15 per cent.

The extraction of silver from Yedras ore by the Russell process is somewhat variable, and depends, outside of changes in the character of the ore, in a great measure on the roasting, as already stated previously. During 1892, the average "apparent" extraction was 87.57 per cent., with a minimum of 80.45, and a maximum of 91.24 per cent. The extraction in the mill, compared with the extra-solution assay-office tests, fell short of the latter by 1.27 per cent.

TREATMENT OF OLD TAILINGS

BY THE RUSSELL PROCESS.

All the tailings which had been produced at the Yedras mill by the old lixiviation-process, before the introduction of the Russell process in 1887, have now been re-treated by the latter. The manipulation of the solutions for tailings is practically the same as in the treatment of ore.

The tailings are brought from the old dumps, where they had been thrown out in former years, and charged directly into the ore-tanks without preliminary treatment. Like the charges of ore, they are leached first with water in order to remove the small percentage of soluble salts which is present, this washing requiring about four hours. A small amount of ordinary solution is then applied, not because this solution extracts anything, but in order to prevent aqueous dilution of the standard extra-solution, of which only a quantity is used sufficient for saturating the charge.

As in treating ore, the standard extra-solution amounts to 13 cubic feet per ton. It is followed by more ordinary solution to extract silver which has been made soluble by the extra-solution, but has not yet passed out with that solution, remaining mechanically held in the charge.

The total amount of tailings from the old lixiviation-process at Yedras which have been re-treated by the Russell process is about 30,000 tons.

Table No. 1801 contains the results of treating tailings for three years.

TABLE No. 1801.

Year.	Value per Ton.	Extraction by Russell Process in Assay Office.	Apparent Extraction by Russell Process in Mill.	Actual Extraction by Russell Process in Mill.
	Ozs.	Per Cent.	Per Cent.	Per Cent.
1888	19.49	62.70	60.14	62.74
1889	17.23	57.20	55.95	60.14
1890	13.46	49.26	48.37	46.84
Av'g's	16.73	56.39	54.82	56.57

Chemicals used per Ton of Tailings.

Year.	Hyposulphite.	Blue-Stone.	Caustic Soda.	Sulphur.	Cost of Chemicals per Ton.	Ounces Silver Extracted per Ton.
	lbs.	lbs.	lbs.	lbs.	Dollars.	Ozs.
1888	2.08	6.32	4.54	3.30	1.45	12.23
1889	1.62	5.11	3.13	2.69	1.12	9.75
1890	1.07	4.06	2.48	1.71	0.84	6.30
Av'ges	1.59	5.10	3.38	2.57	1.13	9.43

The value of the product from the treatment of old tailings at Yedras, by the Russell process, averaged about 7,000 ounces in silver per ton.

B. The Russell Process compared with Amalgamation.

§ 1804. Results at the Sierra Grande, Cusihuiriachic, and Blue Bird mills :

TABLE No. 1804.

Name of Mill.	Ore From:	Value of Roasted Ore Minus Soluble Salts.	Per cent. of Apparent Extraction.			
			In Assay-Office.		In Mill.	
			By Ord. Sol.	By Extra-Sol.	By Amalgamation.	By Russell Process.
*Sierra Grande.	Sierra Grande Mines.	19.0	71.0	81.5	70.6	83.0
†Cusihuiriachic.	San Antonio Mine.	50.2	85.0	89.2	86.3	88.5
	San Miguel Mine.	58.5	89.0	91.9	87.5	90.4
‡Blue Bird.	Blue Bird Mine.	?	74.0	88.9	72.2	86.5

* In these experiments only from 3 to 4 tons of ore were amalgamated for several days, and the results were compared with the regular work of lixiviation on ore of the same character and value.

† In these experiments 100 tons of ore were treated by amalgamation, and the same quantity by the Russell process.

‡ The above figures represent average results in treating about 2900 tons of ore by the Russell process, compared with amalgamation of ore roasted at the same time. The statistics are taken from a pamphlet on the Russell process at the Blue Bird mill by C. A. Hoyt. The value of the ore is not given; but so far as I know, it was low-grade, and contained about 15 to 20 ounces of silver per ton. The Blue Bird mill, Butte, Montana, has a crushing-capacity of 150 tons per day; the ore is roasted in two Stetefeldt furnaces and one Howell furnace. Amalgamation having become unsatisfactory on account of increased baseness of the ore from the lower levels of the mine, a lixiviation-plant was added to the mill, in 1889, after a car-load of ore had been successfully

treated by the Russell process in the experimental plant at the Ontario mill.

Below is a partial analysis of Blue Bird base ore, made from a six months' average battery-sample.

	Per cent.
Iron	3.74
Manganese	5.21
Copper	0.20
Lead	4.22
Zinc	12.80
Sulphur	5.00
Silica	64.40

It is not explained why an ore with 5 per cent. sulphur, and without lime, after roasting with 13 to 15 per cent. salt in a Stetefeldt or Howell furnace, should give a silver-chlorination of only 74 per cent. Most likely, the capacity of these furnaces was over-taxed owing to insufficient draught, or the ore was not allowed to remain long enough on the cooling-floor to have its chlorination raised by heap-roasting. Hoyt mentions that the ore was wetted down while still red-hot after remaining from 18 to 20 hours on the cooling-floor. This time, according to my experience, is insufficient for finishing the silver-chlorination of an ore containing nearly 13 per cent. of zinc. If the ore had been left for two days on the cooling-floor, and had not been wetted down while red-hot, I have no doubt that the extraction both by amalgamation and by the Russell process would have been materially increased.

After writing this, I find a correspondence in the Engineering and Mining Journal in reference to this matter as follows: The comparative run between amalgamation and lixiviation (the results of which are found in Table No. 1804) was finished about six months before the Blue Bird mill was finally closed down. During these last six months amalgamation alone was used. We have recently learned from Mr. Hoyt that during this time the roasted ore was cooled without wetting down while red-hot. The result was an apparent extraction by amalgamation of about 90 per cent. What the extraction by the Russell process would have been on such ore is not known, but the difference in its favor would, of course, have been much smaller than in the comparative run above mentioned.

APPENDIX.

Cost of treating Tailings at the Blue Bird Mill.

After the Blue Bird mine failed to supply ore to the mill, 15,797 tons of tailings from amalgamation were treated by the Russell process. The cost of treatment, according to official figures furnished by Mr. C. A. Hoyt, was as follows:

Hauling tailings to the ore-tanks	$0.35
Labor in lixiviation	.35
Chemicals	.43
Pumping water	.06
Superintendence, etc.	.32
Total	$1.51

This does not include the royalty paid to the Russell process company. Mr. Hoyt has not seen fit to state the value of the tailings and the percentage of extraction. Treatment of tailings was suspended on account of the low price of silver.

§ 1805. *Results at the Marsac and Ontario Mills, Park City, Utah, on Daly Ore, obtained in* 1893–94.

While in all previous paragraphs statistics were based on commercial assays only, Table No. 1805 contains the figures for both commercial and corrected assays, and the apparent as well as the actual extraction is given, the latter based on the value of the roasted ore.

The lixiviation-runs were made at the experimental plant of the Ontario mill, and the amalgamation clean-ups were made at the Marsac mill.

TABLE No. 1805.

Lixiviation and Amalgamation of Daly Ore.

	Weight of Ore. Tons.	Value of Roasted Ore minus Soluble Salts.		Value of Tailings.		Apparent Mill Extraction based on		Actual Extraction by Clean-up based on	
		Comm. Assay. Ounces.	Correct. Assay. Ounces.	Comm. Assay. Ounces.	Correct. Assay. Ounces.	Comm. Assay. Per cent.	Correct. Assay. Per cent.	Comm. Assay. Per cent.	Correct. Assay. Per cent.
Lixiviation.	19.0	33.23	34.88	2.6	3.0	92.2	91.4	100.1	94.9
	25.5	39.43	42.77	3.0	3.6	92.4	91.6	100.7	92.8
	32.9	39.55	42.71	2.8	3.4	92.9	91.8	100.0	92.6
	31.0	37.39	40.38	2.95	3.6	92.1	91.1	102.7	95.1
	26.8	36.81	39.57	2.1	2.4	94.3	93.4	99.8	92.8
	36.9	40.38	43.00	2.75	3.4	93.2	92.1	100.6	94.5
	27.4	37.43	40.60	3.1	3.6	91.7	91.1	99.4	91.6
Amalgamation.	703	35.30	38.91	3.2	3.6	91.0	90.7	104.1	94.4
	795	35.70	38.90	3.2	5.4	89.1	86.2	95.5	87.5
	831	37.10	40.90	4.9	6.4	86.7	84.2	92.9	84.4

Taking the averages of these figures we find that the actual extraction, by clean-up based on the commercial assay, was 100.5 per cent. in lixiviation, and 97.5 per cent. in amalgamation; based on the corrected assay it was 93.5 per cent. in lixiviation, and 88.7 per cent. in amalgamation.

CHAPTER XIX.

LIXIVIATION AT THE MARSAC, ASPEN AND MOUNT CORY MILLS.

§ 1901. *Lixiviation at the Marsac Mill, Park City, Utah.*[*]

The Marsac mill treats the ore from the Daly mine, which is on the same vein as the Ontario. An analysis of the ore for 1894 is not extant; but one was made from an average of all battery-samples for 1891, and gave the following result:

SiO_2, 76.60; CaO, 1.32; Zn, 5.30; Pb, 3.50; Fe, 1.65; Cu, 0.39; S, 0.70 per cent.

Antimony, arsenic, etc., were not determined.

The character of the ore has not materially changed since 1891, except that it is slightly baser. The average value of the ore, for 1894, was 36.98 ounces silver per ton.[†] Most of the silver is contained in decomposed fahlore.

After passing the revolving driers, ore and salt are crushed separately by stamps.[‡] No. 20 wire-screens are used on the ore-battery, and No. 16 on the salt-battery.

The ore and salt mixture, containing 8.2 per cent. salt, is roasted in a Stetefeldt furnace fired by gas from a Taylor gas-producer.

From 0.25 to 0.5 per cent. of sulphur are generally mixed with the pulp before roasting to improve the chlorination of the silver, and to increase the rate of lixiviation in the ore-tanks.

After discharging from the furnace and dust chambers, the roasted ore remains banked on the cooling-floor for 24 to 30 hours. It is then spread out and wetted down while still red-hot. Daly ore is one of the few ores that will bear this treatment without perceptible injury.

Before charging the ore-tanks (of 17 feet diameter and 8 feet 7 inches depth), they are filled with 45 inches of water. By adopting this method the quantity of wash-water precipitates, mentioned in § 1401, has been considerably reduced, and the rate of lixiviation has been increased.

---

[*] From notes furnished by W. G. Lamb, and from statistics obtained through R. C. Chambers, President and Superintendent of the Daly Mining Company.

[†] The small amount of gold in the ore is not determined in the regular pulp-assays.

[‡] The crushing-rate is 2.2 tons per stamp in 24 hours for ore.

As soon as the water has risen near the top of the tank, or about 5 or 6 hours before the charging of 72 tons of roasted ore is finished—which takes about 26 hours—base-metal leaching is commenced, sufficient water being run on top of the ore so that, when charging is ended, the water just comes to the surface of the ore after leveling it. In this way time is gained in washing, the water in the charge is gradually cooled, and less of the white precipitate is formed.

The charge of 72 tons stands about 90 inches high in the tank after base-metal leaching has commenced, leaving 12 inches space above the ore.

After 80 inches of wash-water have been used, 30 inches of so-called extra wash-water follow, *i. e.*, water in which 0.1 per cent. of blue-stone has been dissolved. Marsac wash-water is nearly neutral—sometimes slightly acid, and sometimes slightly alkaline, depending upon changes in the character of the ore, principally its contents in lime.

All lixiviation-solutions are used warm, *i. e.*, they are heated in the storage-tanks to about 38° C. The principal reason for this is that the rate of lixiviation is thereby materially increased. With warm solutions it is from 4.35 to 3.5 inches, decreasing as lixiviation of a charge progresses; formerly, with cold solutions, it averaged a little less than 2½ inches.

The concentration of the stock-solution is kept at 2.0 to 2.2 per cent. in hyposulphite.

The concentration of the extra-solution is 0.75 per cent. in blue-stone, and the proportion of blue-stone to sodium hyposulphite is about 1 to 3.

The following is the method of applying the solutions:

	Inches.	Cubic Feet, per Ton.
First wash-water	80	21.0
Extra wash-water	30	7.8
First ordinary	120	31.5
First extra	20	5.25
Second ordinary	40	10.5
Second extra	20	5.25
Third ordinary	100	26.2
Second wash-water	60 or more.	15.7 or more.

When about 40 inches of first ordinary have sunk through the charge, weak solution makes its appearance at the outlet of the tank, which is turned into the launder leading to the weak solution precipitating-tanks. The concentration of the solution in hyposulphite is now determined from time to time by the iodine-test, and when it reaches from 0.6 to 0.7 per cent. it is turned into the launder leading to the solution precipitating-tanks. This generally occurs when about 20 inches more solution have passed through the charge. These inches are noted as a guide for drawing the line when the second wash-water is applied, so that the volume of the stock-solution remains constant. The total time consumed in working a charge is as follows :

	Hours.
Charging	26
First wash-water and extra wash-water	24
Ordinary and extra-solutions	80
Second wash-water	26
Sluicing tailings	1½
Total	157½*

During 1894, wash-water and weak solution were still precipitated together with sodium sulphide. In January, 1895, separate precipitating-tanks were put up for the weak solution, which is now treated like the strong solution. Separate precipitation of lead is practiced at the Marsac mill, using Solvay soda as precipitant, and silver is precipitated with sodium sulphide made in the mill. Wash-water and weak solution are precipitated completely, and the strong solution closely.

During 1894, the base sulphides, *i.e.*, the precipitates from wash-water and weak solution, averaged 3350.5 ounces in silver and 3.084 ounces in gold, per ton. After starting the weak solution precipitating-tanks, samples of base wash-water sulphides assayed 870 ounces silver and 0.70 of an ounce gold, per ton, and the weak solution sulphides assayed 11,624 ounces silver and 9.0 ounces gold.

The carbonates from weak solution assayed 208 ounces silver and 0.32 of an ounce gold, per ton, and only 1.6 per cent. lead.

An analysis of these carbonates has not yet been made.

---

* The time varies more or less and is not the same with every charge.

All precipitates, after collection in filter-presses and drying, are ground in a Brückner ball-mill through a No. 40 wire-screen for sampling. Carbonates and base sulphides are shipped and sold to smelters. The regular, high-grade sulphides are refined at the mill by the sulphuric acid process.

The value of the roasted ore averaged 33.36 ounces silver, per ton.

It contained 7.24 per cent. soluble salts, making the calculated value 35.96 ounces, per ton.

The average value of the tailings was 3.328 ounces, per ton, showing an apparent extraction of 90.75 per cent.

Assay-office tests showed an average chlorination of the roasted ore of 91.52 per cent.; the extra-solution extracted in the assay-office 92.71 per cent. of the silver.

The actual extraction of silver at the Marsac mill is an unknown quantity since, like in all mills working ores from their own mines, accurate weights of the dry raw ore and of the roasted ore are not taken.

The silver in the product of the mill was distributed as follows:

			Value per Ton.	
	By Weight.	Silver.	Ounces.	
In:	Per cent.	Per cent.	Silver.	Gold.
Regular Sulphides	40.27	77.11	9,827.4	11.22
Base Sulphides	22.44	14.65	3,350.5	3.08
Carbonates	30.97	3.68	609.6	0.63
Mill cleanings	5.34	1.00	1185.1	1.00
Bullion from Amalgamation*	0.98	3.56	.638 fine	.0006

The carbonates contained 25 per cent. lead.

Chemicals consumed per Ton of Ore.

	Pounds.
Sodium hyposulphite	7.86
Solvay soda	5.68
Blue-stone	5.84
Sulphur	3.66
Caustic soda	5.51

* Some ore was amalgamated for experimental purposes.

*Cost of Lixiviation per Ton of Ore.**

	Dollars.
Labor	0.799
Steam and power	0.086
Chemicals	0.951
Assay-office	0.095
Sundries	0.021
Repairs	0.053
Refinery	0.325
Bullion expense	0.185
Expense sulphides sold	0.310
	2.825

In the annual report of the Daly Mining Company, 1894, the total cost of reduction, per ton of ore, is given at $7.30. This includes marketing of bullion, base sulphides and carbonates, but is exclusive of the royalty paid to the Russell Process Company.

The Marsac mill treated, in 1894, 22,138 tons of ore.

APPENDIX.

Wages paid at the Marsac Mill; Cost of Coal and Chemicals.

Lixiviation foreman	$200 00	per month.
Men on ore-tanks, 8 hour shift	3 00	
Helper " 10 "	2 75	
Men charging ore-tanks, 8 hour shift	3 00	

Chemicals.

Sodium hyposulphite	$2 76	per 100 pounds.
Blue-stone	4 84	"
Solvay soda	2 47	"
Sulphur	1 87	"
Caustic soda	4 41	"

Coal.

Rock Springs, $4.40; Coalville, $3.15 per ton.

§ 1902. *Lixiviation at the Aspen Mill.*†

The Aspen mill treated, in 1892, 30,856 tons of ore of the following average composition:

SiO_2, 21.66; $BaSO_4$, 20.92; CaO, 10.99; MgO, 4.24; Fe, 10.02; Zn, 2.85; Pb, 2.27 (humid assay); Cu, 0.16; S, 8.10 per

* This does not include the royalty paid to the Russell Process Company.

† Extracts from the annual report of the General Manager, W. S. Morse.

cent. (not including S in $BaSO_4$), and 27.9 ounces silver per ton. 2665 tons of ore contained gold; average, 0.085 ounces per ton.

It is not expressly stated whether all the lime and magnesia was present as carbonate; but assuming this was the case, 10.99 $CaO = 19.64$ per cent. $CaCO_3$, and 4.24 $MgO = 9.01$ per cent. $MgCO_3$.

The 8.10 per cent. sulphur are not sufficient to convert all the iron, zinc, lead, and copper into pyritic minerals; hence, a portion of these base metals, especially the iron, must have existed in the form of oxides. In what combinations the silver occurred, the report does not say.

The ore, after drying in shelf-driers, was crushed by stamps (850 pounds) either through a No. 24 or a No. 30 wire-screen, and the average crushing-rate was 4.09 and 3.80 tons respectively per stamp in 24 hours.

The salt was crushed by stamps (650 pounds) through a No. 12 wire-screen, the crushing-rate averaging 1.37 tons per stamp in 24 hours.

The ore was roasted in a Stetefeldt furnace, fired by gas, with 11.23 per cent. salt (determined by solubility of the Stetefeldt feeder sample) at the rate of 90 to 92 tons in 24 hours. The roasted ore, after discharging, was allowed to remain on the cooling-floor for 4¼ days, and then charged cold, and without wetting down, into the ore-tanks. Charge-samples contained, on an average, 14.08 per cent. soluble salts.

The assay-office extraction by ordinary solution (chlorination) was 78.98 per cent., and that by extra-solution 89.78 per cent., on a silver-value of 24.62 ounces per ton. Chlorinations and extraction by extra-solution were highest in the month of August, namely, 84.85 and 92.08 per cent. respectively.

The weight of a normal charge for an ore-tank, of 17 feet diameter and 9 feet depth, was 60 tons. Before charging, the tank was filled with water (in which, occasionally, chemicals were dissolved) to a depth of 60 inches, sufficient to completely saturate the ore. After starting the washing, a second volume of wash-water would follow.

Chemicals—soda-ash, sodium bi-carbonate, acid, etc.—were sometimes used in the first wash-water, especially blue-stone in case the roasted ore was alkaline in consequence of the presence of caustic lime. As a rule, sufficient pyritic ores were added in roasting to obtain an acid or at least a neutral wash-water. In spite of this, however, it would occur that the wash-water became alkaline at the end of base-metal leaching.

The phenomenon of the "going back" of chlorinations of the washed ore has already been fully discussed in § 522.

After washing, 50 inches (=one volume) of solution would be sufficient to saturate the charge.

With acid ores the ordinary and extra-solutions were, as a rule, applied as follows:

3 Volumes of warm ordinary of about 1.8 per cent. concentration in hypo.

1 Volume of extra-solution, averaging 0.5 per cent. blue-stone.

1 Volume of warm ordinary.

1 Volume of extra-solution.

2 Volumes of warm ordinary.

Then followed the second wash-water, the line or dividing-point between solution and wash-water being drawn when about 70 inches of wash-water had been applied.

This treatment varied as to quantity of solutions applied, and strength of extra-solution used. Sometimes a 3d extra and a 4th ordinary became necessary. On an average, 6½ volumes of ordinary, and 2½ volumes of extra-solution were consumed.

The average rate of lixiviation was 13 inches, which, after washing, was generally cut down to 10 inches.

The average time of lixiviating a charge was about 100 hours, including stops of every kind.

The average time of sluicing a charge was 3¾ hours. The more lime present, the longer it took for discharging a tank. With low lime, 60 tons of tailings could be sluiced in 45 minutes; with a charge high in lime, as much as 8 hours were consumed.

The tailings were sampled as described in § 1303. Below are the averages of tailings-samples.

	Ounces Silver per ton.
Top-tailings	4.04
Bottom-tailings	4.86
Average-sample	3.79

The latter sample is used for figuring the apparent extraction. Besides these, the following samples were taken:

Washed ore sample, taken after first wash-water had been applied.

First ordinary sample, taken after the first ordinary had done its work.

Preliminary tailings-sample, taken before commencing sluicing.

A comparison between the apparent extraction in the mill, and the assay-office tests, has already been given in § 522. It remains to say that the actual mill-extraction of silver in sulphides (as per corrected mill-assay) was 94.21 per cent. against an apparent extraction of 86.74 per cent., a difference of 7.47 per cent. This discrepancy is due to the fact that the ore-assays were commercial and not corrected assays. All the above figures refer to roasted and not to raw ore.

The average gold-extraction was 58.72 per cent.

In regard to precipitation of the solutions, Mr. Morse says: Up to June, sodium sulphide made in the mill was used exclusively, and the cost per ounce of silver precipitated ran from 2.42 to 3.45 cents. Since June, both mill-made sodium sulphide and Na_2S crystals were used, the cost of precipitation varying from 2.21 to 3.84 cents per ounce of silver. The consumption of hypo. is undoubtedly increased by the use of Na_2S crystals. Careful experiments showed an increased cost of hypo. of about 7 per cent., or 2.6 cents per ton of ore. On the other hand, the use of Na_2S crystals increased the grade of sulphides in silver about 14 per cent., which made a saving of 13 cents per ton of ore in freight and treatment charges.

As near as I can determine, there was a saving of about 10 cents per ton of ore by using the commercial Na_2S in place of sodium sulphide manufactured in the mill.

During the latter half of the year, lead was precipitated separately from the solutions with soda-ash.

All waste-water from wash-water and weak solution precipitating-tanks was collected in large tanks and allowed to settle there completely. These tanks were partly filled with scrap-iron; weak sodium sulphide solution was also added to the waste-water.

Product.

The silver in the product of the mill, sulphides and lead carbonates, was distributed as follows:

In :	By Weight. Per cent.	Silver. Per cent.	Assay-Value per ton. Silver. Ounces.	Assay-Value per ton. Gold. Ounces.
Solution-Sulphides	70.8	80.3	3790	1.009
Wash-water Sulphides	17.4	17.1	3277	0.552
Waste-water Sulphides	4.1	1.6	1268	0.285
Lead Carbonates	7.7	1.0	432	0.177
			and 43.1 per cent. lead.	

Chemicals.

The chemicals consumed per ton of ore were as follows:

	Weight. Pounds.	Cost. Dollars.
Hyposulphite	9.98	$0.3742
Blue-stone	13.45	0.6331
Soda-ash	3.97	0.1383
Sod. bi-carbonate	0.07	0.0040
Sulphuric acid	0.86	0.0276
Sulphur	3.84	0.0793
Caustic soda	4.95	0.2413
Na_2S crystals	10.07	0.3629
Total	47.29	1.8607

Cost of Lixiviation.

The cost of lixiviation proper was, per ton of ore, as follows:

Labor	$0.6448
Repairs	0.0326
Power	0.0065
Fuel	0.0609
Laboratory Work	0.3153
Chemicals	1.8607
Freight and Refining Sulphides	1.3605
Total	$4.2813

Total Cost of Mill Treatment.

The total cost of mill-treatment, per ton of ore, was as follows:

General Expenses, including Office-Labor and Expense, Insurance, Taxes, etc.		$0.8079
Unloading and Sampling Ore		0.3262
Drying Ore and Salt:		
Fuel*	$0.1322	
Labor on Driers	0.1502	
Labor, transferring Ore from Ore-Beds to Driers	0.3006	
Repairs, etc.	0.0412	$0.6242

* Coal at $3.00 per ton.

Crushing Ore:		
Power*	$0.0975	
Labor	0.3708	
Repairs	0.1100	$0.5783
Roasting:		
Labor	0.3200	
Fuel	0.1819	
Repairs, etc.	0.0579	
Salt	0.8719	1.4317
Cooling-Floor Labor		0.4909
		$4.2592
Cost of Lixiviation, as above		4.2813
Total cost		$8.5405

The above cost is figured on the dry weight of the ore and may seem high, considering the favorable local conditions at Aspen. It is due to the peculiar character of the ore, requiring an abnormally high consumption of chemicals. The large amount of blue-stone consumed produced sulphides unusually low in silver, and made precipitation and treatment of sulphides very expensive when figured on the cost of reduction per ton of ore.

At this mill the ores were purchased outright, which required the sampling and holding of each lot of ore until settlement was made with the owner. This necessitated the handling of ore twice, and the transfer from bedding-floor to mill. This extra expense, amounting to $0.3262 for sampling, and $0.3006 per ton of ore for handling and transferring to mill, should be omitted in comparing cost at the Aspen mill with cost at mills treating their own ores.

§ 1903. *Lixiviation at the Mount Cory Mill, Nevada, in 1884-'85.*†

The ore treated at the Mount Cory mill contained, in a clean quartz gangue, on an average about 6 per cent. sulphurets, zinc-blende, galena, pyrites and fahlore, and 32.5 ounces silver and 0.09 of an ounce gold, per ton. It was more or less variable in character and value, as shown by the following analyses, representing extremes in composition:

* The power is furnished by a Pelton water-wheel.

† From notes of the General Manager, Albert Arents.

Partial Analyses of Mount Cory Ore.

	Class I. Per cent.	Class II. Per cent.
Zn	2.66	1.12
Cu	0.40	0.29
Pb	2.10	2.00
Fe	0.30	0.88
S	2.30	2.06
Ag	53.5 ounces.	25.5 ounces.

After drying the ore in Stetefeldt shelf-driers, it was broken in a Brodie crusher and pulverized fine by 3 sets of 26-inch Krom rolls. The revolving Krom screens were covered on the outside with No. 18 slot-punched sheet-iron. From 5 to 6 per cent. salt was added to the dry ore when dumping it into the self-feeders of the Brodie crusher. This plant had a crushing-rate of 72 tons in 24 hours.

Furnaces of the Brückner type, holding charges of 5 tons, were used for roasting. They had been constructed according to designs of Mr. Arents, and were 6½ feet in diameter next the fire-place, 5½ feet in diameter at the flue-end, and 18 feet long. The time of roasting a charge was quite variable, and was determined by the character of the ore. Ores of Class I were generally roasted in 8 hours; those of Class II required from 12 to 17 hours for good chlorination. The roasted ore was discharged into bins underneath the furnaces and remained there, red-hot, until the next charge was ready to draw. It was then spread on the cooling-floor and wetted down. In this condition it remained banked on the cooling-floor 12 hours longer before it was transferred to the ore-tanks.

The chlorinations of the charge-samples were, as a rule, from 89 to 90 per cent.

A normal charge for an ore-tank, of 11 feet diameter and 3 feet 5 inches depth, consisted of two furnace-charges, or ten tons original weight of the raw ore. The ore was charged moderately warm, still slightly steaming, and the first wash-water was applied from below. As soon as the wash-water leaving the tank showed a specific gravity of 5° B., it ceased to contain any silver and was run to waste. Washing was then continued until a sample indicated 0° B.

The average time of base-metal leaching was 4½ hours; 27 inches or 213 cubic feet of water were used for a charge showing a rate of lixiviation of 6 inches. The wash-water was neutral and extracted only about one ounce of silver per ton of ore. This was due to the low percentage of salt used in roasting. Although the ore was roasted in furnaces of the Brückner type, the "going back" of chlorinations of the washed ore was observed. Chlorination-tests of the washed ore were not made from all charges, and I give below a few examples only.

Time of Roasting. Hours.	Calculated Value of Charge-sample. Ounces per ton.	Chlorination of Charge-sample. Per cent.	Chlorination of Washed Ore. Per cent.	Chlorination "Gone Back." Per cent.
8.	24.0	90.0	80.0	10.0
	24.9	89.6	82.4	7.2
	24.9	89.6	79.1	10.5
14.	37.3	87.7	84.5	3.2
	51.9	90.8	85.4	5.4
	46.2	89.2	82.3	6.9

The wash-water was allowed to drain off as much as possible before commencing lixiviation with ordinary solution, which was exclusively used in this mill. It was now found advantageous to cut down the rate of lixiviation to 3 inches. About 270 inches of solution, or 2130 cubic feet, were used for a charge of 10 tons, and the time consumed in lixiviation averaged 96 hours. The concentration of the stock-solution in hyposulphite was from ¼ to ¾ per cent. Owing to this low concentration, and using the solution without heating it, the amount of lead dissolved was a minimum. All weak solution was run into the wash-water precipitating-tanks.

Tailings were sluiced out through a central discharge in the bottom of the tank; their average value was about 3.5 ounces per ton.

Sulphuric acid was added to the wash-water before precipitation with calcium sulphide in order to keep zinc in solution.

The wash-water sulphides assayed, on an average, 693 ounces silver per ton, and contained 38 per cent. copper.

From the hyposulphite solution lead was first precipitated by caustic lime, which was gradually added until the solution showed a faint alkaline reaction with litmus paper. The lead-precipitate contained 40 per cent. lead, and 605 ounces silver, per ton. Silver was now precipitated from the decanted solution with calcium sulphide in the usual way. All stirring was done by hand. The solution sulphides were not assayed.

All precipitates were gathered in centrifugal machines and the resulting cakes dried in a small reverberatory furnace.

The lead-precipitate and the base sulphides were shipped and sold to smelters; the solution sulphides were treated at the mill by roasting in a reverberatory furnace and melting the roasted product with iron and borax in black-lead crucibles. The resulting bars were from .800 to .900 fine in silver, and about .001 fine in gold. About 3⅓ per cent. of the silver was obtained as a by-product in matte, assaying 1800 ounces in silver per ton; it was sold to smelters.

The actual results of the mill, per ton of ore, are given as follows:

		Ounces Silver.	Per cent.
	Extraction	25.11	77.35
Loss	*in Tailings*	3.50	10.78
	in Roasting	3.85	11.87
		32.46	100.

Starting with the roasted ore, the extraction was 87.8 per cent., although the chlorinations in the assay-office averaged from 89 to 90 per cent. This difference was caused by the "going back" of chlorinations of the washed ore.

Cost of Treatment per Ton of Ore.

		Dollars.	Dollars.
Drying:	Labor	0.273	
	Wood*	0.108	0.381
Crushing:	Labor	1.121	
	Wood	0.794	1.915
Roasting:	Labor	0.758	
	Wood	0.983	
	Salt	1.320	3.061
Cooling:	Labor	0.545	0.545
Lixiviation:	Labor	0.400	

*The very low consumption of only 0.0216 cord of wood for drying, per ton of ore, is due to the fact that Mount Cory ore contained from 3 to 4 per cent. moisture in a clean, hard quartz gangue, free from clay.

*Chemicals:**	2.6 lbs. Hypo	$0.171	
	5¾ lbs. Sulphur	0.338	
	3¼ lbs. Lime	0.032	
	1⅕ lbs. Acid	0.060	$1.001
Handling Precipitates and Melting:†			
	Labor	0.212	
	Coke and Charcoal	0.080	
	Flux and Iron	0.030	
	Crucibles	0.050	
	Wood	0.083	0.455
Sundry Expenses, Repairs, etc.:			
	Labor	0.848	
	Wood	0.062	
	Supplies	0.423	
	Freight	0.220	1.553
Lubricants			0.148
Coal oil and Light			0.074
General Expenses‡			0.300
	Total		9.433

Cost of Materials and Wages at the Mount Cory Mill, in 1884–85.

Wages:	Dollars.
Chief Engineer	6.00
Fire-men	4.00
Men on Rolls	4.50
Men on Ore-tanks	4.00
Roasters	4.00
All other men	3.00

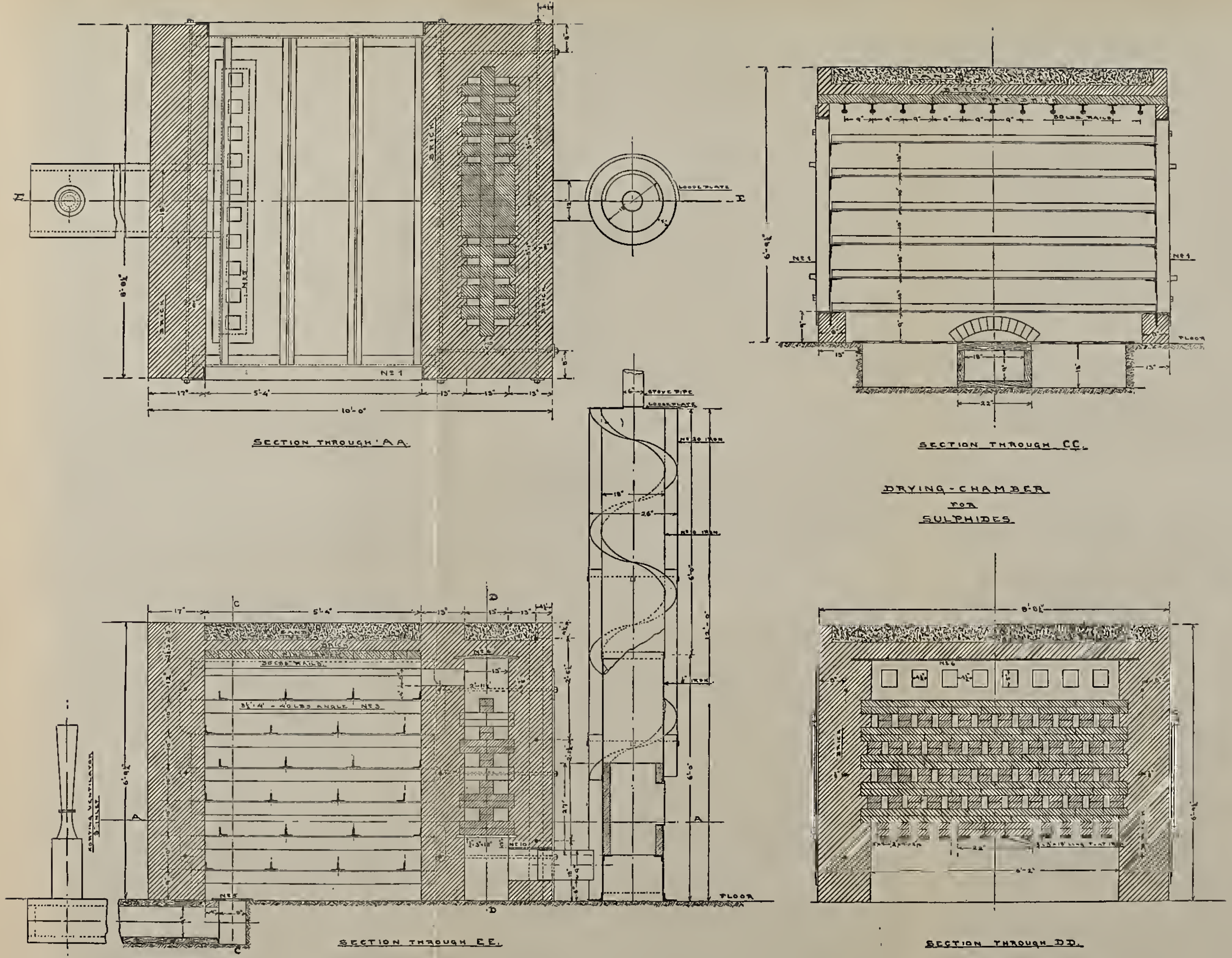
Materials:		
Salt	22.00	per ton.
Wood (nut-pine)	5.00	per cord.
Sodium Hyposulphite	6.42	cents per pound.
Sulphur	5.58	" " "
Sulphuric acid	5.00	" " "
Caustic lime	1.00	" " "

---

* The decimals of the pounds given are only approximate; the dollars are correct.

† The cost of shipping bullion and disposing of lead-precipitates, etc., is not given.

‡ One half of the general expenses, salaries, etc., were charged to the mine.



SECTION THROUGH AA.

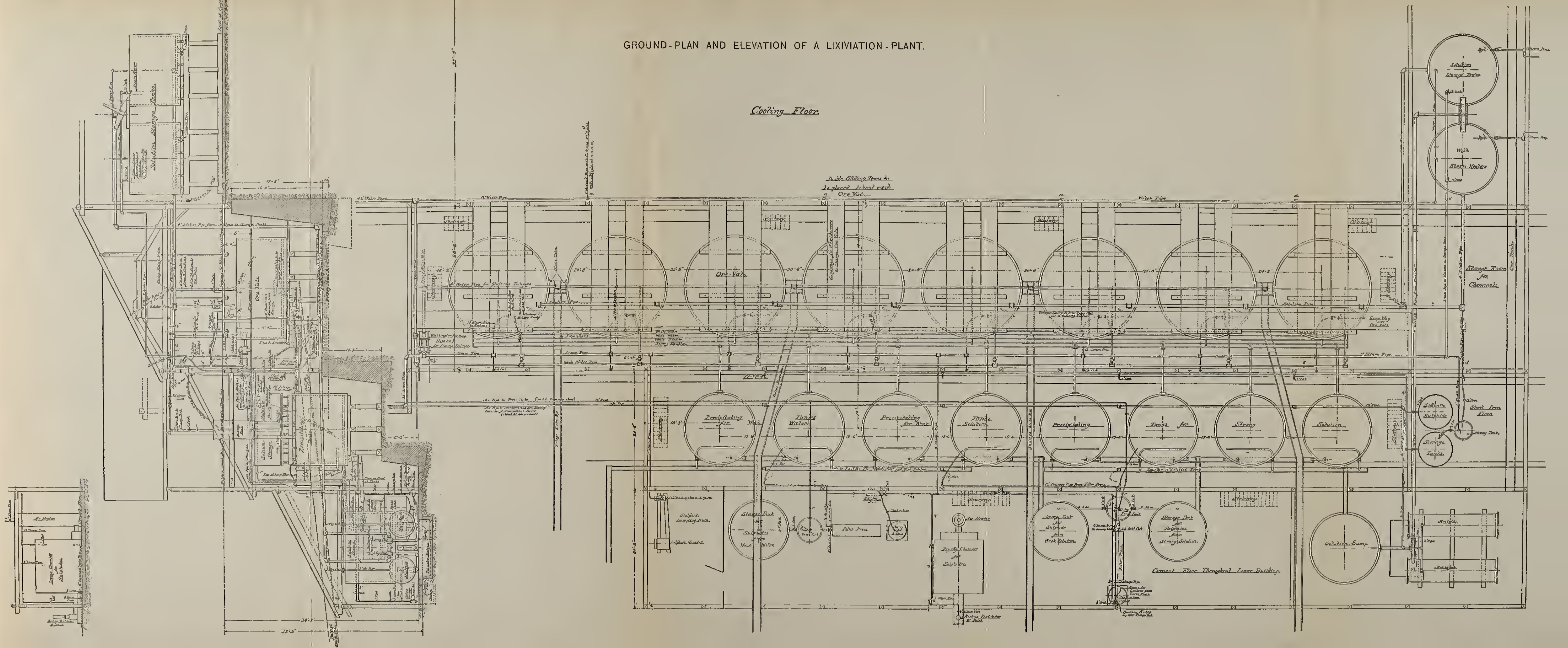
SECTION THROUGH CC.

DRYING-CHAMBER
FOR
SULPHIDES

SECTION THROUGH EE.

SECTION THROUGH DD.

GROUND-PLAN AND ELEVATION OF A LIXIVIATION-PLANT.


Cooling Floor.
Double Sliding Doors to be placed behind each Ore Vat.
Water Pipe
Ore-Vats.
Stairway
Storage Room for Chemicals.
Car Track
Solution Pipe
Gang Way over Ore Vats.
3" Steam Pipe
Solution Storage Tanks
With Steam Heaters
Solution Storage Tanks.
Level of Ceiling
17-8"
16-0"
6'
Precipitating for Wash Water Tanks
Precipitating for Weak Solution Tanks
Precipitating Tanks for Strong Solution.
Sodium Sulphide Storage Tanks.
Sheet Iron Floor
Sulphide Sampling Room.
Sulphide Grinder
Storage Tank for Sulphides from Wash Water
Filter Press
Air Heater
Drying Chamber for Sulphides.
Storage Tank for Sulphide from Weak Solution.
Storage Tank for Sulphides from Strong Solution.
Cement Floor Throughout Lower Building.
Solution Sump
Montejus
Drying Chamber for Sulphides
Air Heater
35'3"
39'7"
15'6"
13'3"
20'8"

INDEX.

A

PAGE
Acid Ores. Definition of 121
“ “ Lixiviation of 144
Air Compressors 115
Alkaline Ores. Definition of 121
“ “ Lixiviation of 145
Argentic Hyposulphite Double-salts 14
Arsenical Alkaline Ores. Definition of 121
“ “ “ Lixiviation of 145
Aspen Mill. Wetting down ore cold and hot 8
“ “ Refining sulphides at the 179
“ “ Lixiviation at the 195
“ “ Chlorinations “going back” at the . . . 63
Asphaltic Covering 116
Assaying of Ores, Tailings, etc. 102
Auric Hyposulphite Double-salts 14

B

Blue Bird Mill. Amalgamation and lixiviation at the . . . 187
Brueckner Pulverizer 160
Bullion-Melting 168

C

Calcium Carbonate. Precipitation of—by sodium sulphide solution . 77
Calcium Chloride. Solubility of—in water 55
Calcium Hyposulphite 10
Calcium Sulphate. Solubility of—in water 56
“ “ “ in brine 57
“ “ “ in sodium hyposulphite. . . 26
Calcium Sulphide. Preparation of 71
Caustics. The effect of—on the solubility of silver-compounds in sodium hyposulphite 21
Caustic Lime. Solubility of—in water 56
“ “ “ in sodium hyposulphite . . . 27
Caustic Soda 11
Chlorinations. The “going back” of 63–202
Cocks 116

PAGE

Construction. General remarks about 117
Copper. Solubility of—in sodium hyposulphite 16
Copper Carbonates. Solubility of—in sodium hyposulphite . . 23
Copper Vessel. Influence of—in lixiviation-tests . . . 96
Cupric Hydroxide. Solubility of—in sodium hyposulphite . . 25
Cuprous Chloride. Solubility of—in sodium hyposulphite . . 25
" " " in brine 57
" " " in water 56
Cuprous Hydroxide. Solubility of—in sodium hyposulphite . . 24
Cuprous Hyposulphite Double-salts 28
Cusihuiriachic Mill. Amalgamation at the 187
" " Lixiviation at the 182

D

Drying-Chamber for Precipitates 114

E

Extra-Solution. Apparatus for circulating 113
" " Composition of—in mill 124
" " Decomposition of—by heating . . . 30
" " " " at ordinary temperature . . 31
" " from Cuprous Chloride 37
" " Circulation of 126
" " Effect of—on raw ores 40
" " " on ores after oxidizing-roasting . . 44
" " " on ores after chloridizing-roasting . 48
" " " on tailings 51
" " Reactions of 29
" " of Standard Composition 28
" " Rules for preparing Standard 124
" " Per cent. of copper sulphate in . . . 126
" " Temperature of 127
" " Volume of—per ton of ore 126

F

Filters for Ore-Tanks 107
Filter-Press 114

G

Gold. Solubility of—in extra-solution 38
" " in sodium hyposulphite 15
Gold Sulphide. Solubility of—in extra solution . . . 40
" " " in sodium hyposulphite . . . 17
Gold. The extraction of—in lixiviation 52

H

PAGE

Heating Stock-Solution. Apparatus for 110
Hyposulphite. Determination of—with iodine 99
" Loss of—in lixiviation 90–122
Hyposulphite Salts. Formation of 72

I

Iodine-Test 99
Iron Mortar. Influence of repulverizing roasted ore in an . . 95

K

Koerting Ejector 112

L

Laboratory Work 93
Launders 109
Lead. Distribution of—in the solution during different periods of lixiviation 146
" Double-salt of—and sodium hyposulphite 14
" Precipitation of—with sodium carbonate 153
" " " with caustic lime 202
" Solubility of—in sodium hyposulphite 17
" " " in extra-solution 38
Lead Antimonate. Solubility of—in sodium hyposulphite . . 26
" " " in wash-water brine . . . 61
Lead Carbonate. Solubility of—in hyposulphite solutions . . 23
Lead Chloride. Solubility of—in water 56
" " " " in brine 57
Lead Hydroxide. Solubility of—in hyposulphite solutions . . 24
Lead Sulphate. Solubility of—in brine 61
" " " in sodium hyposulphite . . 25
Lixiviation of Raw Ores 143
" " Roasted Ores 144
Lixiviation-Plant 105
" " Cost of 118
Lixiviation-Solutions. General rules for handling the . . . 129
Lixiviation-Tests. With Ordinary Solution 95
" " " Extra-Solution 97

M

PAGE

Marsac Mill. Amalgamation and lixiviation at the . . . 189
" " Lixiviation at the 191
" " Refining sulphides at the 158–180
Mount Cory Mill. Lixiviation at the 200
Montejus 111–113

O

Ore-Tanks. Charging of the 128
Ordinary Solution. See stock-solution.

P

Pipe-Lines 115
Precipitating-Coefficients 81
Precipitation of Metals from Hyposulphite Solutions . . . 69
Precipitating-Tanks for Wash-Water 110
Press-Tank 114

R

Rate of Lixiviation 131
" " " Apparatus for increasing the . . . 112
Refining Sulphides 157
Roessler Converter 161

S

Salt. Determination of—in crushed ore 94
Sierra Grande Mill. Amalgamation at the 187
" " " Lixiviation at the 182
Silver. Distribution of—in the solution during different periods of lixiviation 146
" Precipitation of—with sodium or calcium sulphide . . 154
" Solubility of—in sodium hyposulphite 15
" " " in extra-solution 37
Silver Antimonate. Solubility of—in brine 61
" " " in sodium hyposulphite . . 18
" " " in extra-solution . . . 40
Silver Arsenate. Solubility of—in brine 61
" " " in sodium hyposulphite . . 18
" " " in extra-solution . . . 40
Silver Carbonate. Solubility of—in sodium hyposulphite . . 23

PAGE
Silver Chloride. Solubility of—in brine 58
“ “ “ in various chlorides . . . 59
“ “ “ in sodium hyposulphite . . 23
“ “ “ in wash-water brine . . . 60
“ “ Influence of calcium sulphate on the solubility of—
in sodium hyposulphite . . . 20
“ “ The same for sodium sulphate . . . 21
“ “ The same for lead sulphate 20
Silver Oxide. Solubility of—in sodium hyposulphite . . . 15
Silver Sulphide. Solubility of—in extra-solution . . . 39
“ “ on top of charge in ore-tanks . . . 133
Silver Tetrathionate 36
Sluicegates 107
Sluicing Tailings. Fire-pump for 115
Sodium Antimonate. Solubility of—in water 57
“ “ in sodium hyposulphite 27
Sodium Carbonate 10
“ “ Solution 127
Sodium Chloride. Solubility of—in water 55
Sodium Hyposulphite 9
Sodium Monosulphide 12
Sodium Sulphide. Preparation of 71
Sodium Sulphide Tanks 110
Sodium Sulphate. Solubility of—in water 55
Soluble Salts. Determination of—in roasted ore . . . 94
Solution-Sumps 110
Stetefeldt Process for Refining Sulphides 158
Stock-Solution. Apparatus for raising 111
“ “ Caustic 123
“ “ Concentration of—in hyposulphite . . . 121
“ “ Preparation of the 121
“ “ Volume of—in the mill 123
“ “ Volume of—used per ton of ore . . . 122
Storage-Tanks for Precipitates 109
Sulphides. Mechanical handling of the 155
“ Refining of 157
Sulphuric Acid Process for Refining Sulphides . . . 179

T

Tailings. Discharging and sampling of 133
“ Treatment of—at Blue Bird mill 189
“ “ at Yedras mill 185
Tanks 105
Testing Ores in the Laboratory 103
Tetrathionates. Conversion of—into hyposulphites . . . 79
Trough Lixiviation 149

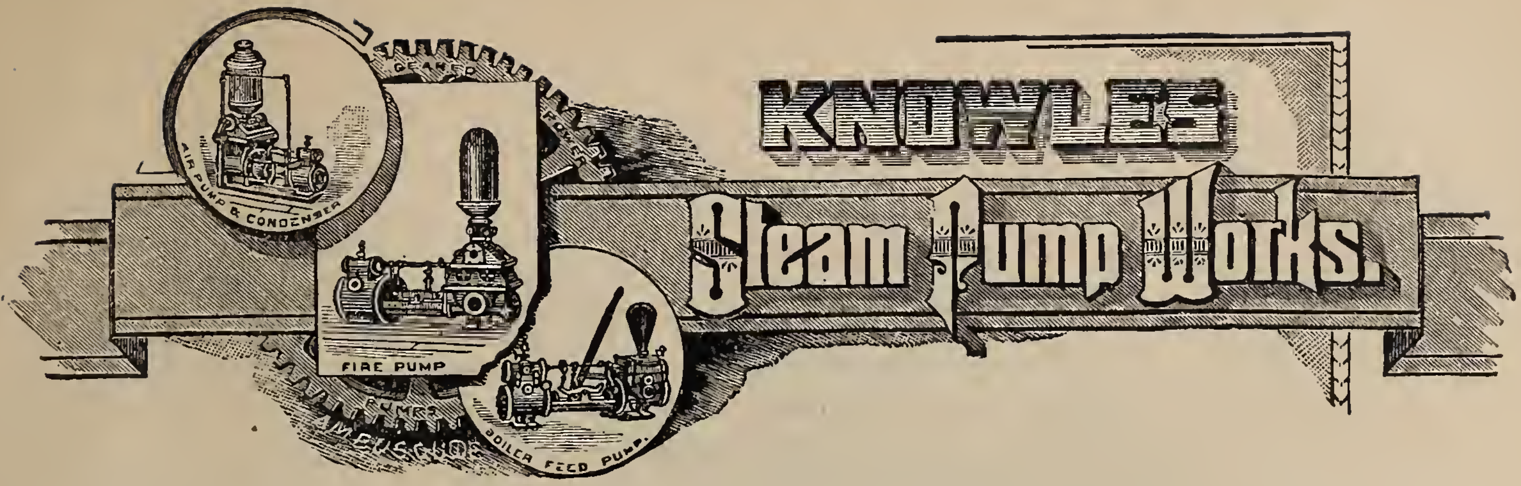
W

PAGE

WASH-WATER. The chemistry of the 54
" " Effect of the—on the "going back" of chlorinations . 63
" " The First 135
" " " " Use of chemicals in the . . . 139
" " Precipitates below filter of ore-tanks . . . 141
" " Treatment of Silver-Bearing 142

Y

YEDRAS MILL. Lixiviation at the 183



KNOWLES Steam Pump Works.

MINING PUMPS A SPECIALTY.

Send for special catalogue of Knowles' Mining Pumps, etc.

PUMPING MACHINERY FOR WATER WORKS

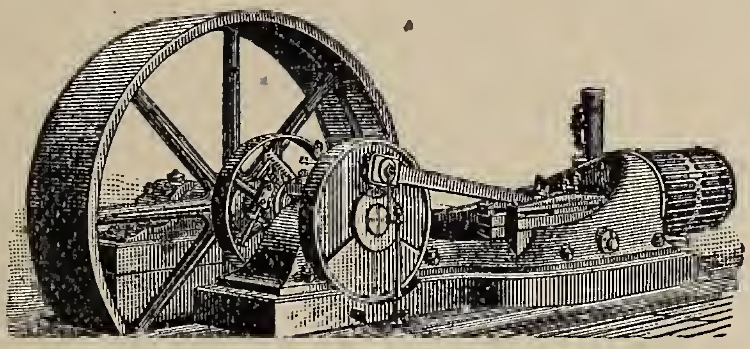
and every other service.

Knowles' Air Compressors for elevating acid solutions by air pressure.

93 Liberty Street, - - - - - New York.

BUCKEYE ENGINE CO.,

Salem, Ohio.



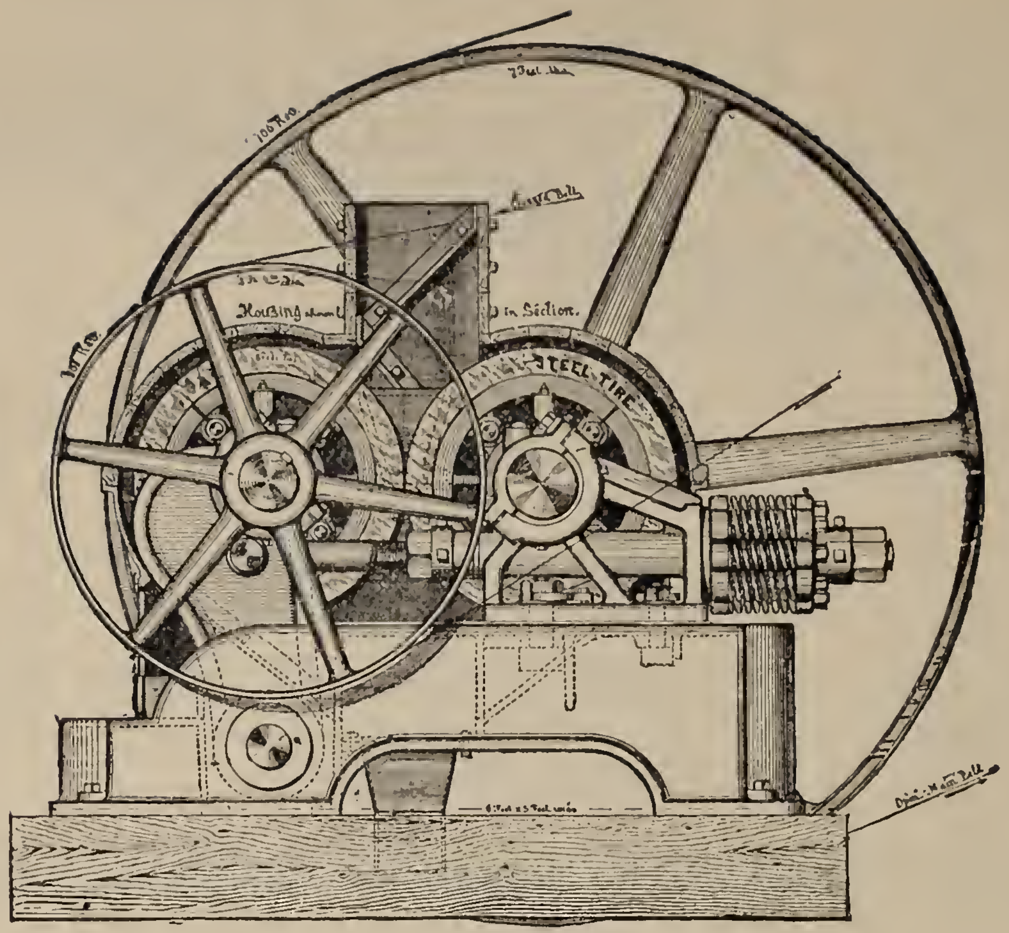
W. L. SIMPSON,

General Eastern Sales Agent,

301 TELEPHONE BUILDING, 18 CORTLANDT ST., NEW YORK.

ENGINES OF ANY SPEED from 75 to 250 revolutions.

ENGINES OF ANY POWER from 13 to 1000 H. P.



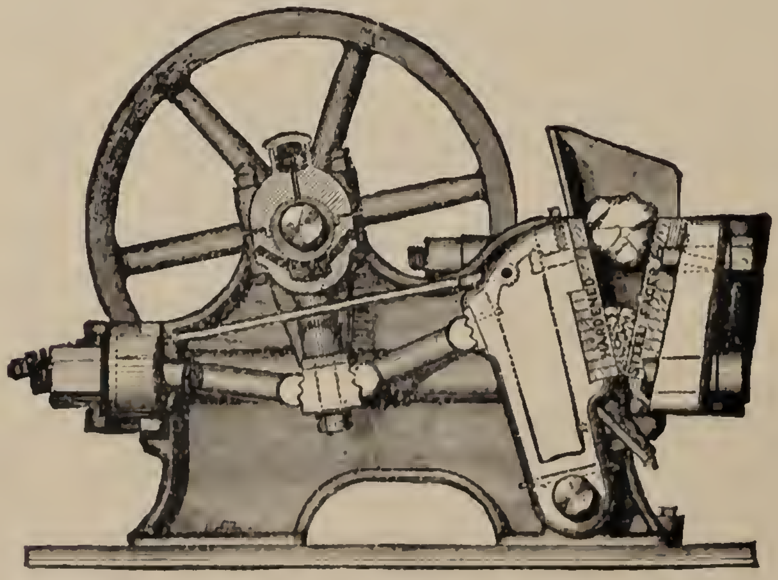
KROM'S PERFECTED STANDARD

STEEL ROLLS and ORE-BREAKERS.

Revolving Screens, Ore-Feeders,
Dry-Kilns and Ore Concentrators.
Laboratory Rolls, Ore-Breakers,
Ore Concentrators and Screens.
Plans for Lixiviation, Concentrating
and Ore-Crushing Works.

S. R. KROM,

151 Cedar Street, New York, U. S. A.



FRASER & CHALMERS,

Fulton and Union Streets, Chicago, Ill.

MINING AND MILLING MACHINERY

OF EVERY DESCRIPTION.

Complete Plants furnished for the systematic treatment of all ores, on latest improved designs, by

LEACHING, AMALGAMATION, CONCENTRATION AND SMELTING.

Improved fast-running ROLLS — all known successful ROASTING FURNACES and dry kilns—CRUSHERS, STAMPS, PANS, JIGS, BUDDLES, FRUE VANNERS and EMBREY CONCENTRATORS.

Huntington Roller Mills, Sturtevant Mills, Water Jacket Furnaces.

Boss continuous process of Amalgamation — Hallidie Wire Rope Tramways.

Corliss Engines and Boilers.

Stetefeldt Roasting Furnaces and Dry Kilns.

REFERENCE TO FOLLOWING PLANTS BUILT:

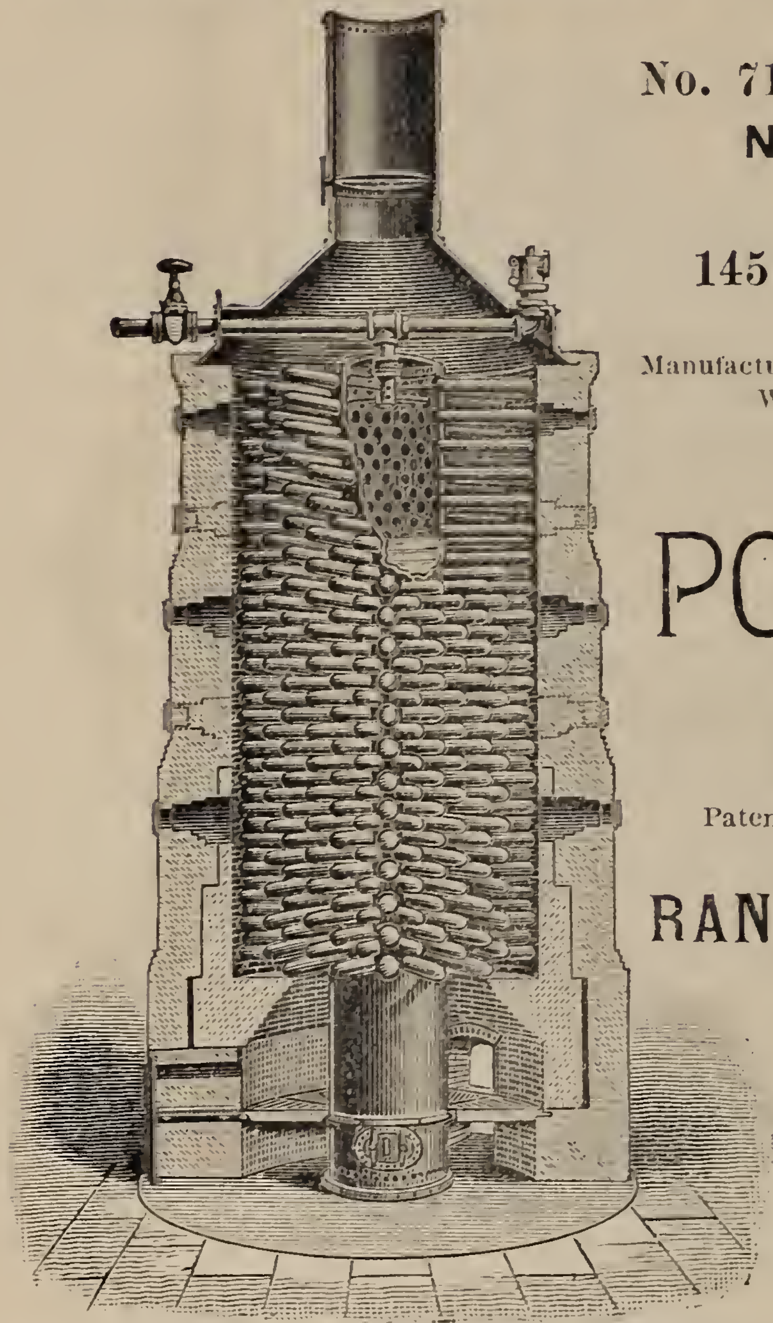
Anaconda Copper Co.,	Mon.,	10	steam stamps.	
Granite Mountain Co.,	"	160	stamps,	silver.
Bi-metallic Mining Co.,	"	40	"	"
Montana Mining Co.,	"	110	"	"
Blue Bird Mining Co.,	"	60	"	"
Empire Mining Co.,	"	60	"	gold.
Homestake Cos.,	Dakota,	550	"	"
Rio del Oro (A),	Arg. Rep.,	25	"	"
" mill (B),	"	25	"	"
" " (C),	"	25	"	"
" " (D),	"	25	"	"
Co Todos Santos,	Chili,	10	"	silver.
Broken Hill Prop.,	Australia,	3	steam stamps.	

Segui Mill,	Uruguay,	20	stamps,	gold.
Gregorio Mini'g Co.,	Uruguay,	40	"	"
Guanandi Mining Co.,	"	20	"	"
Uruguay Gold Mg. Co.,	"	30	"	"
Société St. Elie	Fr'nch G'yana,	20	"	"
Cie La Mana	" "	20	"	"
El Callao,	Venezuela,	60	"	"
V'nezuel'n Austin,	Venezuela,	40	"	"
El Choco,	"	20	"	"
La Union,	"	20	"	"
Backus & Johnston, Peru, Concentrator.				
Paarl Pretoria, Africa, Gold Mill,				

BRANCH OFFICES:

NEW YORK, 2 Wall street.
LONDON, 23 Bucklersbury, E. C.
DENVER, 1316 18th street.
SALT LAKE CITY, 7 W. 2d S. st.
MEXICO, 11 calle de Juarez, Chihuahua City.
Also at JOHANNESBERG, South Africa, and LIMA, Peru.

THE HAZELTON BOILER CO.,



General Office:
No. 716 EAST 13th STREET,
NEW YORK, U. S. A.

Branch Office:
145 Broadway, New York,
Room 28.

Manufacturers and Sole Proprietors of that type of Water-Tube Boiler frequently called

THE PORCUPINE BOILER,

Patented in the United States and Foreign Countries.

RANKIN, BRAYTON & CO.

Proprietors of Pacific Iron Works,
SAN FRANCISCO, CAL.

Manufacturers of General Mining Machinery and Sole Manufacturers of this Boiler for California, Nevada, Idaho, Oregon, Washington, Alaska, Arizona, Lower California, British Columbia and West Coast of Mexico.

Standard 215 Horse Power Stationary Boiler, with one-half of Brickwork, Smoke-Hood and Smoke-Stack removed.

The Largest Volume of Very Dry Steam, with Least Consumption of Fuel.

Admirably Adapted to the Use of Any Refuse Combustible Matter.

ABSOLUTELY SAFE
Under Unusually High Pressures.

LIGHT IN WEIGHT.
VERY DURABLE.

No Brick Chimneys Required. Accessible in Every Part for Interior and Exterior Examination and Cleaning. Easily Repaired.

THE HENDEY AND MEYER ENGINEERING CO.,

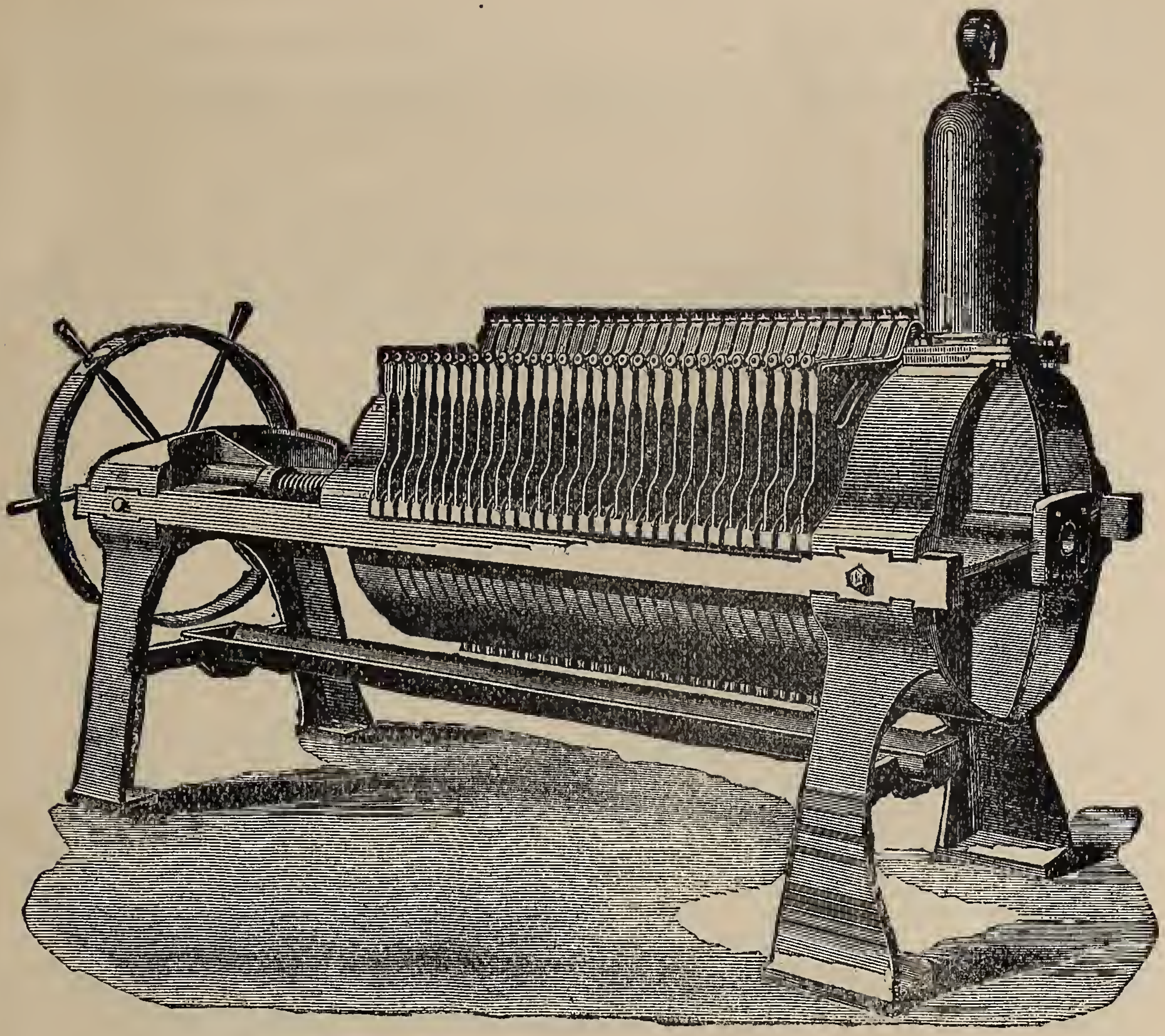
DENVER, COL.

Manufacturers of General Mining Machinery and Sole Manufacturers of this Boiler for Colorado, Montana, Wyoming, Utah, New Mexico and East Coast of Mexico.

We are prepared to erect Plants of Boilers of any desired capacity, and solicit inspection of those now in operation. Catalogues giving full information sent free on application.

CORRESPONDENCE SOLICITED.

THE "JOHNSON" FILTER-PRESS.



JOHN JOHNSON & CO.,

FRANKLIN SQUARE,

NEW YORK.

MAKERS AND PATENTEES OF FILTER-PRESSES

For perfect Filtration of Liquids; separating, collecting and pressing the Solids therefrom. Also arranged for Lixiviating or Leaching soluble matter from the solids collected, while in the Filter-press, in one operation. Also makers of Forcing Receivers for Filling Filter-presses without the use of pump.

☞ SEND FOR CATALOGUE. ☜

SULPHUR.

CRUDE SULPHUR, FLOWERS OF SULPHUR,
GROUND SULPHUR, VIRGIN ROCK,

FOR

ACID WORKS, SHEEP DIP, SULPHURIZING VINES

AND FOR

Preparation of Sodium and Calcium Sulphide

USED IN THE

LIXIVIATION-PROCESS.

DICKERT & MYERS SULPHUR CO.,
P. O. Box 768, - - - Salt Lake City, Utah.

THE SOLVAY PROCESS CO.,

Syracuse, N. Y.,

MANUFACTURERS OF

SODA-ASH, BI-CARBONATE

AND

CAUSTIC SODA.

We guarantee test and quality of every shipment. The best and most uniform Soda in the market.

WING & EVANS, SOLE AGENTS,
54 William Street, - - - New York City.

TN
770
S84

Stetefeldt, Carl August
The lixiviation of silver-ores with hyposulphite solutions 2d ed.

Mining

PLEASE DO NOT REMOVE
CARDS OR SLIPS FROM THIS POCKET

UNIVERSITY OF TORONTO LIBRARY

[redacted]